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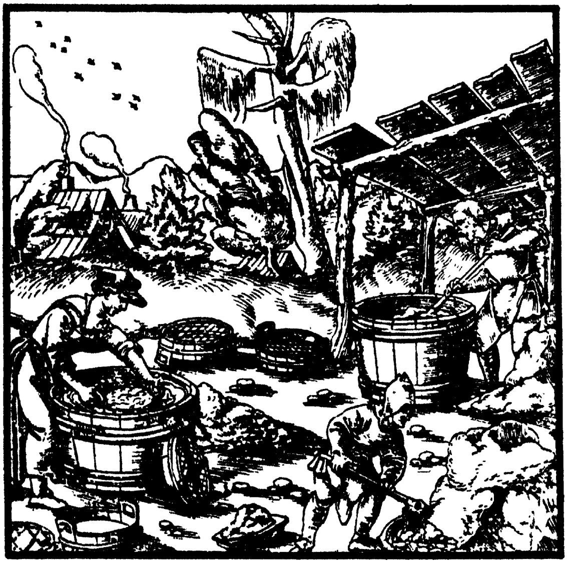
THE
DRESSING OF MINERALS



A Sixteenth Century Hand-Jig.

(*See page* **264.**)

THE

DRESSING OF MINERALS

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WITH OVER 400 ILLUSTRATIONS

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PREFACE

MY object in producing this book is to fill a very definite gap in technological literature. There are numerous excellent works on the subject of Mining and fully as many on that of Metallurgy, but no English work has yet appeared dealing with the present subject, covering as it does ground common to both of the above branches of Technology, nor is there indeed any modern Continental work that can be said to bring it at all up to date.

Minerals are not, as a general rule, suitable, exactly as mined, for the purposes of the smelter or for use in the Arts and Manufactures, and they usually have to undergo a series of operations, at times extremely simple, but at others also highly elaborate, in order to fit them for such use, and it is to this series of operations that the term "Dressing" is applied. The object of the present work is to give an account of the theory and practice of the Dressing of Minerals, which will, I hope, prove useful to the miner or metallurgist who desires to understand the principles upon which this art is based, as also to the manufacturer who supplies the necessary appliances, and above all to the student who is preparing for either of the above professions. Although portions of this subject have been dealt with to some extent in a number of works, its practical and economic importance fully justifies the production of a text-book devoted to it alone. In the present work I have made somewhat of a new departure, and have followed the method that I have been applying for the last twelve years in my Lectures on the Dressing of Minerals, which I have regularly treated as an independent subject, inasmuch as I have disregarded the time-honoured division, which would make separate branches of the Dressing of Ores, and of the Cleaning of Coals. These have hitherto been looked upon as different subjects, and have been discussed independently: seeing, however, that both depend upon identical principles, and are often carried out in appliances that differ only in trifling details, I hold that each is capable of throwing light upon the other, and I hope that the simultaneous treatment of both in the present work will bring out clearly the underlying fundamental principles, and will afford a comprehensive view of the application and scope of the entire subject.

Although the literature on the Dressing of Minerals is scanty, a very large amount of practical work has been done upon it, notably by the manufacturers of the machines and appliances employed, and I am greatly indebted to such manufacturers, in all parts of the world, for drawings and descriptions of their productions, and for information concerning them, which they have most liberally put at my disposal. I have endeavoured in every case to acknowledge fully this assistance in the pages of the book itself; here

I need only add, that wherever I have described any appliance as made by some particular firm, I do not thereby mean to imply that this make is necessarily superior to similar machines made by others, my selection having been guided mainly by the desire to present well-marked typical examples of the various appliances. A point of considerable difficulty arises with respect to appliances that have been only recently introduced; as a rule, whenever the machines appeared to be designed in accordance with sound principles, I have referred to them briefly, and have stated that they had been in use for too short a time to enable a fair judgment to be formed concerning them.

My thanks are also due to a number of Technical Institutions for their kind permission to reproduce illustrations and information contained in their respective publications; these, also, I have endeavoured fully to recognise. I have further to record my thanks for assistance in different sections, received from my colleagues, Professor R. M. Ferrier, M.Sc., M.Inst.C.E., of University College, Bristol, Professor H. Stroud, D.Sc., and Mr J. H. Morrow, D.Sc., of this College, and finally to my assistant, Mr H. Dean, M.Sc., A.R.S.M., to whom I am particularly indebted for the compilation of the Index.

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Armstrong College,
Newcastle-on-Tyne.
February, 1909.

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THE DRESSING OF MINERALS

CHAPTER I.

GENERAL CONSIDERATIONS.

THE Dressing of Minerals (German *Aufbereitung,* French *Préparation mécanique*) is a term that is intended to include all the series of operations that intervene between the extraction of a mineral from its natural deposit and its production in a condition ready for sale or for further use in the arts or manufactures; it practically covers the whole series of operations which have for their object the separation of a mass of mixed minerals into its mineralogical constituents, or the cleaning of one mineral constituent from the accompanying mineral or minerals which may be looked upon from the technical point of view as the impurities of the former. It is therefore essentially a branch of Technology. The limits of a scientific subject are capable of clear and exact definition, and its scope is easily fixed, but this is far from being the case with a technical art, the boundaries of which are merely decided by expediency or prescribed by custom. Accordingly we find that it is all but impossible to say exactly where the operations included in the term "Dressing of Minerals" begin, or where they leave off, and we find them encroaching on the one hand upon the domain of the miner, and on the other overlapping those of the metallurgist and manufacturer. It will therefore be necessary to decide at the outset within what range we will limit our subject; the most satisfactory method perhaps is to take the mineral as it leaves the mine. The greater proportion of all minerals got by mining operations, properly speaking, is drawn out of the mine from shafts or adits in mine tubs or waggons, and dressing may be taken to commence at the point where this mineral is tipped out from these waggons, any tipping arrangements that may be necessary being looked upon by some as forming part of mining proper, and by others as belonging to dressing. In many instances the mineral is not drawn out in waggons, but is hoisted up shafts in skips or kibbles, the contents of which are either transferred direct to waggons similar to those referred to above, or else are emptied into bins, and from these into waggons, or again, instead of waggons in either case, the buckets of an aerial rope-way may be made use of. By analogy with the former case it will again be convenient to take about the same starting point, namely, when the mineral is emptied out from these waggons or buckets, although it does not leave the shaft bottom in them. The series of dressing operations may be looked upon as completed when the mineral, cleaned and ready for market, is again loaded into the waggons in which it is to be conveyed from the dressing establishment; here again an exact limit is difficult to fix, and it might fairly be argued that operations incidental to loading the mineral on board ship might fairly be included, although they but rarely are.

The range of operations that are embraced by the subject of dressing, as thus empirically defined, is very variable. It may be as well to premise that the word mineral is used not in the restricted sense to which it is confined by strict mineralogical definition, but with the loose conventional meaning that is attached to it in mining, commercial and legal phraseology, and that it means more nearly the objective of mining than anything else. A mineral in this sense very rarely or never occurs in nature in the pure state, but always more or less intermixed with other worthless or injurious constituents, which have to be removed to render the mineral proper marketable at its full value. It is proper to remark that we occasionally have to separate minerals, each of which is valuable by itself, but may be injurious to the other; thus a mixture of zinc blende and galena may contain too much zinc to allow of its being smelted for lead, and too much lead to allow of its being smelted for zinc; yet when separated, the blende and galena would be valuable ores of the respective metals. Hence the amount of dressing required may vary within the widest limits, fro nothing at all to what is practically complete metallurgical treatmen For instance, a good deal of coal is sold as "through and through" "run of mine"; that is to say, the coal as it comes from the mine simply loaded into railway cars and is then an article of commerc the same is the case with certain iron ores. At the opposite en of the scale stands, for example, a low-grade gold quartz, which woul be utterly valueless if it had to be carried as such to any considerabl distance from the mine in which it is got; the only valuable ingredient in it may be the small proportion of gold that it contains, and the ore is subjected to a series of operations including breaking, sorting, screening, crushing, concentrating, and possibly even chemical after-treatment, before the marketable portion of it is extracted from the worthless residue. In this case dressing and metallurgical operations pass into one another so imperceptibly that it is scarcely possible to say where the line should be drawn. The same difficulty is met with in the dressing of crude tin ore, the final stages of which even include furnace operations, such as calcination, which are usually looked upon as purely metallurgical.

Again, the proportions of valuable and worthless ingredients in the mass of mineral as raised from the mine vary within very wide limits. On the one hand we may have, as already pointed out, coals and iron ores which are saleable as they stand, whilst with the vast majority of the coals raised in this country over 95 per cent. of the material brought to bank is valuable mineral. On the other hand we find, for example, auriferous gravel worked by what is practically an operation of dressing, aided by some chemico-metallurgical methods, in which the amount dressed out is about one part in twenty-five million, or diamondiferous blue ground, in which about one part in forty million is separated out by a series of true dressing operations.

The nature of the mineral substance itself has further a very definite influence on the dressing operations required to fit it for sale. All minerals that are got by mining operations may be conveniently classified in the following five groups:

I. Fuels,
II. Ores,
III. Salts,
IV. Gems,
V. Rocks.

Group I contains, in the first place the various varieties of coal, asphalts, bitumens and other solid hydrocarbons, mineral oils, etc., whilst sulphur may also conveniently be included. It is advisable to restrict the term dressing to the treatment of solid mineral products, and to exclude the various operations of refining oil (as well as the distillation of oil-shale), which, although they might fairly be considered here, are so entirely different from the general scope of dressing operations that they are best omitted for purely practical considerations.

Group II includes not only compounds of the heavy metals, but also such as occur in the native state, so that in this sense gold quartz, for example, will be looked upon as an ore of gold. On account of the mode of their occurrence in deposits, only a small proportion of which is often constituted by the ore which it is sought to obtain, this group furnishes by far the largest number of examples of dressing practice. It has very often been the custom to treat the dressing of groups I and II as entirely distinct subjects, and to speak of the washing of coals as a branch quite separate from the dressing of ores; seeing, however, that the principles upon which the operations depend are absolutely identical, that the machines employed are always similar and sometimes quite the same, and that the objects of the operations are identical, it appears better to treat coal-washing and ore-dressing as one and the same subject, any difference of treatment due to the modes of occurrence and physical characteristics of these two groups being duly explained in its proper place.

Group III is taken to include haloid salts, sulphates, carbonates, silicates, etc. of the alkalies and alkaline earths, whilst compounds of alumina, such as felspar, for example, are also best considered along with them. The word metal will therefore be restricted to the heavy metals, or, in other words, to those metals which the chemist recognises as being precipitated in the first three groups of chemical separation, alumina alone being excluded. In speaking therefore of an ore deposit consisting say of galena and fluorspar, it will be correct in the above sense to speak of the former as the metallic and the latter as the nonmetallic portion, although it is quite true that fluorspar is a compound of the metal calcium just as much as galena is of the metal lead; the above division is, however, practically a very convenient one.

Group IV contains a number of minerals differing widely from each other in many respects, but having the characteristics of hardness and transparency in common; it also includes what are often spoken of as the semi-gems, such, for example, as opal, lapis lazuli, turquoise, etc., which are opaque, but are nevertheless prized as precious stones on account of the beauty of their colour and their rarity.

Group V consists of rock masses which are more often quarried than mined in the usual sense of the word. Their treatment will not be considered here, although the dressing and splitting of slates, the cutting of basalt or granite paving sets, the squaring of blocks of freestone, and even the polishing of marbles and similar stones might possibly be looked upon as forming a portion of the subject. As the meaning that is here given to the term dressing of minerals is, however, quite empirical, it is best to exclude all these operations, which may in a sense be looked upon as forming the basis of separate trades, and to restrict the term dressing within the limits above indicated, namely, to the separation of mineral constituents from each other.

Finally, the nature of dressing operations in any given case will depend upon the mode of occurrence of the minerals. Given that a mineral mass consists of two constituents which have to be separated, it will obviously make a great difference whether these constituents occur loose, as in gravels, or closely coherent, as in veinstuffs, and whether they occur individually in large masses or in very fine particles. The sizes of the particles of mineral in a deposit may vary within very wide limits indeed; at the one end of the scale may be placed such occurrences as coarsely crystalline pegmatite, as an excellent example of which the stanniferous granite rocks of Dakota may be quoted, with individual crystals weighing several hundredweights, and at the other end of the scale may be placed certain gold ores in which the gold occurs in a state of subdivision such that the particles are invisible except under a tolerably powerful microscope.

It will be obvious, from what has been said, that the dressing of a mineral mass may under certain circumstances consist of a series of very complex operations, whilst in other cases it is an extremely simple matter. A satisfactory classification of the entire subject is accordingly scarcely possible; the method here adopted will be to describe in the first place the various individual operations and the machinery employed in their execution as so many independent units, and then to *point out how these various individual appliances and processes are combined to produce the desired result in each particular case.* This method of presenting the subject has the further advantage that it accustoms the student to the great elasticity of methods, which is one of the characteristics of dressing practice, and enables him to see that there are usually several methods available for attaining the end aimed at; as a matter of fact it very often happens that several different combinations of the same, or even of different, operations will often produce in practice results so nearly identical that even the experienced engineer will hesitate which to select, and that local custom will often be found to play an important part in the final decision.

Ore-dressing operations depend upon differences in the properties of the minerals to be separated from each other, and are simpler in proportion as these differences are greater and more strongly marked. The properties of which advantage is taken for these purposes may be:

I. Mechanical: brittleness or toughness or friability; hardness or softness; form, size, or structure.

II. Physical: colour and lustre; specific gravity; magnetic susceptibility; electric conductivity; surface adhesion to liquids.

III. Chemical: solubility; action of heat and certain chemical agents.

The processes to be employed depend upon which of these properties can be made available, their importance in this respect being very unequal; some are almost universally applicable, others only in very special cases. The individual processes admit of no very exact classification, and can be combined together in almost infinite variation, even the order in which they are employed being sometimes a matter of indifference. For the purpose of facilitating their study they are best perhaps grouped as follows:

I. Processes depending on mechanical properties:

(*a*) Simple volumetric sizing, i.e. separation of larger particles from smaller.

(*b*) Picking or sorting, i.e. separation of valuable from worthless minerals, or of different varieties of valuable minerals from each other, by the eye alone; in this form and structure are very important, but the physical properties of colour and lustre are scarcely less so, whilst the trained hand of the sorter even uses specific gravity to aid him in his selection.

(*c*) Breaking or Comminution; this may be either a preliminary to picking or to more elaborate processes; it may either precede or follow sizing, or the two operations may alternate, even several times in some cases. Breaking may be subdivided into:

(*α*) Coarse breaking $\begin{cases}\text{(i) by hand,}\\ \text{(ii) by machinery.}\end{cases}$

(*β*) Fine breaking or crushing.

Hand breaking is very often combined with picking.

(*d*) Washing to remove adhering mud, dust, or dirt, sometimes merely as a preliminary to picking, sometimes as a complete process.

II. Processes depending on physical properties:

(*a*) Separation according to specific gravity. This forms by far the most important portion of the whole subject of dressing, and may be subdivided as follows:

(*α*) Separation in fluids of specific gravity intermediate between those of the bodies to be separated, the fluids being generally at rest.

(β) Hydraulic separation in currents of water; the great majority of dressing processes are included under this head.

(γ) Pneumatic separation.

(b) Magnetic separation, applicable when one of the minerals to be separated is either naturally more magnetic than the other, or else is readily rendered magnetic, e.g. by heating.

(c) Electric separation, applicable when one of the bodies to be separated is more readily electrified or a better conductor of electricity than the other.

(d) Separation by surface tension, depending on the greater adhesion of fluids or gases to the surfaces of certain of the mineral particles.

III. Processes depending on chemical properties.

These are very rarely used; they may consist of:

(a) Solution in water, followed by crystallisation, evaporation, or precipitation.

(b) Solution in acids.

(c) Solution in mercury.

(d) Special chemico-metallurgical operations.

This last group of processes requires little more than a bare mention, as they are scarcely included within the limits of subject as here defined.

The material which the dresser has to treat depends upon the nature of the mineral deposit to be exploited and upon the method of exploitation adopted. The study of each of these respective conditions forms an independent subject, the former, which is strictly speaking a portion of economic geology, being generally spoken of as the science of Mineral Deposits, the latter constituting the Art of Mining properly so called. The student must be referred for information upon these subjects to the works dealing specifically with them; a general acquaintance at any rate with their essential portions will here be presupposed. In one sense dressing operations may commence to a certain extent underground; thus when a mineral deposit is very variable in character, if it is mined by any method that necessitates the leaving of pillars, care will be taken to so lay out the work that the poorest portions shall be left, whilst the richer shall be extracted. Or when the entire mass of the deposit is excavated, barren portions will often be left behind in the mine and packed in the spaces from which the mineral has been removed, and only those portions sent out for treatment which promise to repay the cost. The question to what extent such underground preliminary sorting should be carried is a complex one, depending upon the character of the deposit, the kind and cost of labour available, the expense incurred in hoisting and after treatment, and a number of similar economic considerations. It has repeatedly happened that the worthless "deads" of one generation have been found capable of being extracted and worked to advantage by another.

The number of mineral substances that may require treating by dressing is very large indeed, and the mineral deposit itself is at times a highly complex mixture of different mineralogical species, this being more especially the case when the contents of mineral veins are being dressed. An engineer in charge of dressing operations ought necessarily to be a competent mineralogist, able to recognise the various minerals, and conversant with their properties. A very great number of these minerals occur but rarely, whilst others are almost invariably present. In addition to true mineralogical species, a number of rocks are likely also to be met with, the general characteristics of which must also be known.

In the following table a list is given of all the more commonly occurring minerals and rocks that are likely to be met with in ordinary dressing operations, together with the more important of their properties which can be made available for the separation of the minerals from each other or from the rocks in which they occur, or with which they may be associated.

GROUP I. *Fuels.*

Mineral	Specific gravity	Hardness on Mohs's scale	Other properties
Bituminous coal	1·2—1·37	0·5—2	Breaks into more or less cubical blocks; brittle
Anthracite	1·3—1·8	2—2·5	Breaks with a conchoidal fracture
Lignite	1·15—1·3	0·5—1·5	Sometimes lamellar and rather friable
Asphalt.....................	0·9—1·8	1—2	Melts about 100° C.
Ozokerite..................	0·8—0·9	0·5—1	Melts about 60° C.
Sulphur	2·1	1·5—2·5	Melts about 110° C.
Graphite	2—2·2	1—2	Very fissile

GROUP II. *Ores.*

Mineral	Specific gravity	Hardness on Mohs's scale	Other properties
Native gold	15·6—19·5	2·5—3	Soluble in mercury; malleable
„ Platinum	17—22·5	4—6	Malleable
„ Silver	10—11	2·5—3	Soluble in mercury; malleable
Silver glance	7·2—7·4	2—2·5	Somewhat sectile
Proustite	5·4—5·6	2—2·5	Brittle
Pyrargyrite	5·7—5·9	2—2·5	Brittle
Kerargyrite	5·3—5·5	1—1·5	
Native copper	8·8—8·9	2·5—3	Malleable
Copper glance	5·5—5·8	2·5—3	
Copper pyrites	4·1—4·3	3·5—4	
Malachite	3·7—4	3·5—4	
Tetrahedrite	4·5—5·1	3—4·5	
Cinnabar	8·9—9	2—2·5	
Galena	7·2—7·7	2·5—3	Cleaves easily into cubical fragments
Cerussite	6·4—6·5	3—3·5	
Anglesite	6·1—6·4	2·5—3	
Pyromorphite	6—7·1	3·5—4	
Bournonite	5·7—5·9	2·5—3	Brittle
Cassiterite	6·4—7·1	6—7	
Stibnite	4·5—4·6	2	
Bismuthite	6·4—7·2	2	
Pyrites	4·8—5·2	6—6·5	
Marcasite	4·7—4·9	6—6·5	
Pyrrhotite	4·4—4·7	3·5—4·5	Brittle, magnetic
Mispickel	6·0—6·4	5·5—6	
Magnetite	4·9—5·2	5·5—6·5	Strongly magnetic
Haematite	4·5—5·3	5·5—6·5	
Brown haematite	3·5—4·4	3—5·5	
Spathic ore	3·4—3·9	3·5—4·5	
Ilmenite	4·5—5	5—6	Sometimes slightly magnetic
Chromite	4·3—4·5	5·5	Sometimes slightly magnetic
Pyrolusite	4·8—4·9	2—5·5	
Manganite	4·2—4·4	4	
Zinc blende	3·9—4·2	3·5—4	
Calamine	4—4·5	5	
Wolfram	7.1—7·6	5—5·5	

GROUP III. *Salts.*

Mineral	Specific gravity	Hardness on Mohs's scale	Other properties
Salt	2·1—2·6	2·5	Soluble in water
Carnallite	1·6	2	Soluble in water
Fluorspar	3—3·2	4	Easily cleavable
Cryolite	2·9—3·1	2·5	
Calcspar	2·5—2·8	2·5—3·5	Very easily cleavable into rhombohedral fragments
Dolomite	2·8—2·9	3·5—4	
Magnesite	2·8—3·2	3·5—4·5	
Witherite	4·3—4·4	3—4	
Gypsum	2·3	1·5—2	Easily cleavable
Kieserite	2·5	2·5	Soluble in water
Barytes	4—4·7	2·5—3·5	
Boracite	2·9—3	4·5—7	
Soda nitre	2·0—2·3	1·5—2	Soluble in water
Apatite	2·8—3·2	4·5—5	
Kaolin (clay)	2·2—2·6		
Felspars	2·5—2·8	6—7	
Micas	2·7—3·1	2—3	Cleave readily into thin plates

GROUP IV. *Gems.*

Mineral	Specific gravity	Hardness on Mohs's scale	Other properties
Quartz	2·5—2·8	7	
Diamond	3—5	10	Easily cleavable
Corundum	3·9—4·2	9	
Emerald	2·6—2·8	7·5—8	
Spinel	3·5—4·9	8	
Garnet	3·1—4·3	6·5—7·5	
Topaz	3·4—3·7	8	Cleaves with moderate ease
Zircon	4·1—4·8	7·5	
Opal	1·9—2·3	5·5—6·5	
Hornblende	2·9—3·7	5—6	
Augite	3·1—3·6	5—6	
Tourmaline	2·9—3·3	7—7·5	

GROUP V. *Rocks.*

Mineral	Specific gravity	Hardness on Mohs's scale	Other properties
Basalt	2·7—3·1		At times somewhat magnetic
Diabase Diorite Gabbro	2·6—3·1		
Syenite	2·7—2·9		
Trachyte Andesite Rhyolite	2·3—2·8		
Granite	2·6—2·7		
Quartz porphyry Felsite	2·5—2·7		
Gneiss	2·6—2·7		
Crystalline schist	2·7—3·1		
Slate	2·5—2·9		
Shale	2·5—2·8		
Clay	1·9		
Earth	1·5—2·0		
Limestone	2·5—2·7		
Marble	2·7—2·8		
Oolitic limestone	2·0—2·4		
Chalk	2·8—2·6		
Magnesian Limestone	2·7—2·9		
Sandstone	2·2—2·6		
Sand	1·5—1·9		
Shingle	1·4—1·5		

CHAPTER II.

VOLUMETRIC SIZING.

Theory of sizing. The sizing of a quantity of broken minerals is the operation of dividing it into two or more parcels according to the sizes of the particles; it may be applied either to mineral as it leaves the mine, in which case the difference in size of the various pieces is due either to their mode of occurrence or to the action of the explosives or other means employed in their extraction, or it may be applied to mineral which has been previously broken or crushed in the course of dressing; it may be a preliminary operation either to breaking or to picking, or it may be an intermediate operation between coarse breaking and fine crushing, or it may form the last stage of the process of crushing, and will then generally be a preliminary to hydraulic, pneumatic, magnetic or some other mode of separation. Sizing may therefore be applied to pieces of minerals of very widely varying dimensions from big lumps measuring say a foot or more in any direction down to the very finest flour; it is obvious that the apparatus employed will have to vary accordingly.

Sizing is always performed by passing the material to be sized over a screen or a series of screens, a screen consisting of a plane or curved surface provided with apertures of various shapes and of the required dimensions; these apertures may be either long slots, or round holes or else square, rectangular, hexagonal, octagonal, or diamond shaped apertures; they may be formed by assemblages of bars, by variously perforated plates, or by gauzes, woven in various ways, and made almost invariably of wire.

The portion that passes through the apertures of a screen is spoken of as the "Undersize" of that screen, whilst the portion, which consists of pieces too large to pass through, is spoken of as the "Oversize" of that screen; the term "Reject" is sometimes used instead of the latter, but the word "Oversize" appears to be preferable.

The theory of sizing is at first sight excessively simple, depending merely on the obvious fact that a particle will drop through a hole larger than itself. In reality however it presents a series of most complicated problems, the solution of which has barely been attempted. Theoretically a particle (say a sphere for the sake of simplicity) should fall through a hole of the same diameter as itself in an infinitely thin plate. Practically, however, a hole through a plate of definite thickness forms a tube and the resistance of the walls of the tube will prevent a sphere of exactly the same size as the tube from falling down it. In order to pass through a screen therefore, a sphere must be of smaller diameter than the holes in the former, and the difference will have to be greatest if the holes are circular, less if they are square or diamond shaped, and least if they are slots long in proportion to their width. The thinner the plate also, the larger the particle that can fall through an aperture in it of given size. Little more than the above vague statement can be given as quantitative determinations of the relative sizes of particles and meshes are still wanting. The shape of the particles to be sized is also a factor of great importance; it is evidently possible that a long columnar piece might pass through a round hole, though its length be much greater than the diameter of the hole, and a flat piece will even more readily pass through a screen consisting of bars, provided that one of its dimensions (its thickness) be less than the space between the neighbouring bars. It is possible in any case to determine a sphere which would have the same weight as the average of the largest pieces that pass through a screen and this may be called their "equivalent sphere." Rittinger has found that, using square meshes, the diameter of the equivalent sphere is equal to

0·73	of the	width	of the	mesh	for spheroidal	fragments
0·87	„	„	„	„	tabular	„
0·80	„	„	„	„	columnar	„

No data are available respecting the ratios of the equivalent spheres of particles screened through apertures of various shapes, but these will obviously be greatest in the case of long slots, less for square, and least for round apertures. In screening coal it has been found in one case that as much small is taken out by bars placed ⅜ inch apart as by round holes 1 inch in diameter punched in an iron plate; the construction of the screens was however different in the two cases.

Sizing is generally performed dry; it is, however, also often done wet, either under still water or in a stream of running water. It must however be borne in mind that although dry screening and wet screening are both quite feasible and extensively used in practice, it is impossible to screen stuff in an intermediate state, i.e. moist. Moist mineral clots together and refuses to screen, and at the same time clogs the screen apertures, especially in the finer sizes; such material can only be sized by either first thoroughly drying it and screening it dry, or else by screening it in water, either flowing or at rest.

It will be seen in the sequel that in a large number of screening appliances, the mineral to be screened is allowed to slide over the surface of the screen. The determination of the angle at which sliding takes place is a matter of some importance, and it is necessary to distinguish between the angle at which a mineral just commences to slide (α) and that at which it continues to slide at a uniform velocity after it has started from rest (ϕ); no piece of the particular mineral can remain at rest upon a metal surface inclined more highly than α, and none can permanently continue in motion under the action of gravity alone, upon one inclined less than ϕ. The following determinations were made upon moderately large rough lumps of the various minerals sliding upon a smooth steel plate.

Mineral	Coefficients of friction				
	α	$\tan \alpha$	ϕ	$\tan \phi$	$\frac{\tan \alpha}{\tan \phi}$
Coal	30°	0·577	26°	0·488	1·182
Quartz	25½°	0·477	20°	0·364	1·310
Magnetite	28°	0·582	18¾°	0·340	1·564
Limestone	31½°	0·613	21°	0·384	1·596
Galena	24°	0·445	15°	0·268	1·660
Zinc blende	25°	0·466	15¾°	0·282	1·652
Fibrous red haematite	25¼°	0·472	16½°	0·296	1·594
Calcspar	27°	0·510	20°	0·364	1·401

Designation of apertures. Slotted apertures are designated by the width of the aperture in the clear, square holes by the side of the square, round ones by their diameter; in the rarer case of diamond shaped holes the lengths of both diagonals must be given. Occasionally, with coarse woven screens, the size of mesh from centre to centre is stated instead of width in the clear. When these dimensions are comparatively large, they are expressed in this country in inches or fractions of an inch; decimals are however much to be preferred. For the finer screens certain trade numbers corresponding to the diameters of needles that will just pass through the apertures are often employed, especially in America; sometimes the number of holes to the square inch, or more rarely, to the linear inch, is given, but this is scarcely even an approximate guide to the real size of aperture, except in the case of wire screens. These trade customs, though very general, are unsatisfactory and the practice in various parts of the world is far from uniform; it is far more convenient to express the widths of all apertures (the minimum width if one dimension is the smaller) in decimals of an inch or in millimetres. On account of the differences in the description of screens when made from perforated metal sheets and when made of woven wires, the two are best considered separately.

Perforated plates. The following table[1] represents the general trade practice in designating the widths of screen apertures, the average number of round holes per square inch being also given:

Needle Number	Diameter of hole		No. of holes to the square inch
	inch	mm.	
1	0·057	1·45	70—100
2	0·048	1·225	100—120
3	0·041	1·05	130—160
4	0·035	0·875	140—160
5	0·029	0·75	160—200
6	0·027	0·675	160—200
7	0·024	0·60	240—280
8	0·022	0·55	240—280
9	0·020	0·50	260—300
10	0·018	0·45	260—300
11	0·016	0·413	300—360
12	0·015	0·375	300—360

In Cornwall an entirely different scale is used, it being there mainly applied to the "grates" of stamp mills. According to Messrs Holman Brothers, Ltd. of Camborne the Cornish practice is represented by the following table:

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[1] From *A Handbook of Gold Milling*, by H. Louis, 3rd edn., p. 139.

Cornish scale	Equivalent needle number
33	1
34	2
35	3
36	4
37	5
37½	6
38	7
39	8
39½	9
40	10
41	11
42	12

The practice of spacing the larger holes varies a good deal; very often holes less than 0·25 inch in diameter are spaced at distances apart equal to their diameters, and holes over 1 inch in diameter at distances equal to their radii, intermediate sizes being spaced proportionately. The thickness of sheet or plate from which these screens are manufactured also varies a good deal, many makers having three grades—heavy, medium and light for each size of hole. The following table shews an average practice for medium weights of metal:

Diameter of hole	Thickness of sheet steel
inch	inch
0·015 to 0·03	0·01 to 0·02
0·03 to 0·06	0·02 to 0·035
0·06 to 0·10	0·025 to 0·05
0·10 to 0·15	0·03 to 0·06
0·15 to 0·20	0·04 to 0·10
0·20 to 0·3	0·05 to 0·16
0·3 to 0·5	0·10 to 0·20
0·5 to 1·0	0·10 to 0·25
above 1·0	0·20 to 0·4

Instead of steel sheets, iron is sometimes used; for small holes closely set, Russian sheet iron is much liked, or copper, brass or gun-metal is at times used instead; when the last named materials are employed, the sheets may with advantage be a shade thicker than in the case of iron or steel. Perforated sheets are invariably manufactured by punching; the result of this process is that the holes, particularly in the case of the smaller sizes, are conical or trumpet shaped as shewn diagrammatically in the subjoined magnified section, Fig. 1. The burr thrown up by punching may either be left on, forming what is known as a burred or indented screen, or it may be ground off, forming a plain screen. In the coarser sizes the pieces are punched clean out, thus making always plain screens, alike on both sides.

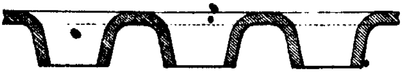

Fig. 1. Burr of screen.

Woven wire screens. Woven wire screens are rarely used for any size of mesh exceeding 2 inches square and are best confined to sizes less than about 1 inch. For finer sizes they are in many respects preferable to punched plate screens, and can be made much finer than the latter. Their material is usually steel wire, but iron, copper, bronze or gun-metal, and brass are also used; brass is especially suitable for all sizes finer than 40 meshes to the linear inch. The following table shews the thickness of wire that is generally used for the manufacture of a screen of any given mesh. There is however much variation in the practice of different makers, whilst each maker will usually furnish one and the same mesh made with various gauges of wire according to the purpose for which it is to be used.

Meshes per linear inch	Clear width of mesh	Thickness of wire
	inch	inch
1	0·75 to 0·89	0·25 to 0·11
2	0·32 to 0·43	0·18 to 0·07
3	0·20 to 0·27	0·13 to 0·065
4	0·15 to 0·20	0·10 to 0·050
5	0·105 to 0·16	0·095 to 0·04
6	0·085 to 0·13	0·08 to 0·035
7	0·07 to 0·11	0·075 to 0·03
8	0·06 to 0·10	0·065 to 0·028
9	0·053 to 0·086	0·058 to 0·025
10	0·051 to 0·078	0·049 to 0·022
12	0·041 to 0·063	0·042 to 0·02
14	0·036 to 0·053	0·035 to 0·018
16	0·034 to 0·046	0·028 to 0·016
18	0·030 to 0·040	0·025 to 0·014
20	0·028 to 0·038	0·022 to 0·012
30	0·019 to 0·024	0·014 to 0·009
40	0·015 to 0·017	0·01 to 0·008
60	0·010 to 0·013	0·007 to 0·004
80	0·011	0·004 to 0·003 Brass
100	0·007	
120	0·005	

Brass wire may be used for any size of mesh, but its use is generally restricted to the finer sizes; it is usual to employ a rather thinner gauge of brass wire than would be the case if steel wire being used.

A modified form of wire gauze is what is known as "Locket work"; this consists of comparatively stout round iron rods, ½ to ⅝ inch in diameter, running across the screen, spaced at 4 inch to 5 inch centres; round these rods are wound wires, running longitudinally, several turns being taken round each rod, so as to leave the requisite width. The arrangement is shewn in Fig. 2; the meshes thus formed are usually 3 to 3½ inches long in the clear, and their widths usually vary between ½ inch and 1¼ inches. Locket work affords a smoother surface than ordinary wire gauze, but is much heavier, and has the disadvantage of presenting a rectangular (long and narrow) aperture. When a wire breaks it has to be drawn out and a new wire put in its place from end to end of the screen. To get over this objection so-called "sectional locket work" has been introduced, in which a wire runs continuously only over one pair of adjoining rods, as shewn in Fig. 3[1]. It does not however make a good screen, and has never been much used. Obviously locket work is only suitable for coarse screens, and should never be used for less than ½ inch mesh.

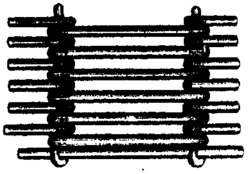

Fig. 2. Locket work.

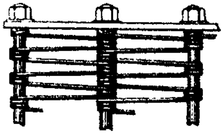

Fig. 3. Sectional locket work.

An attempt has recently been made by the Institution of Mining and Metallurgy to devise standard screen meshes, those selected being square meshes of woven wire, in which the width of aperture between the wires is equal to the diameter of the wires, so that the width of mesh is known when the number of meshes to the linear inch is given. Such screens are useful for laboratory tests, so as to secure uniformity, but are useless for practical screening.

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[1] *Trans. Fed. Inst. Min. Eng.*, "Improved Coal Screening and Cleaning," by T. E. Forster and H. Ayton, Vol. I. 1889–90, p. 83.

SCREENS.

The complete screening appliance consists of the screening surface (bars, perforated plates, woven wire, etc.) and of the framework that carries the former. For the purpose of the present work, screens will be classified into :

1. Fixed screens.
2. Moving screens
   (*a*) in which the screening bars alone move,
   (*b*) in which the entire appliance moves.

1. **Fixed Screens.** This is the simplest form of screening appliance, and is still largely employed, consisting simply of a screen set at a suitable angle so that the mineral to be screened can just slide down the inclined surface whilst the finer particles fall through the screen. These screens are nearly always made of bars arranged longitudinally, and held by bolts at suitable distances apart ; the frame usually consists of a couple of planks set on edge on either side of the bars ; these planks are best lined with sheet iron to avoid undue wear. The following are types of this form of sizing appliance.

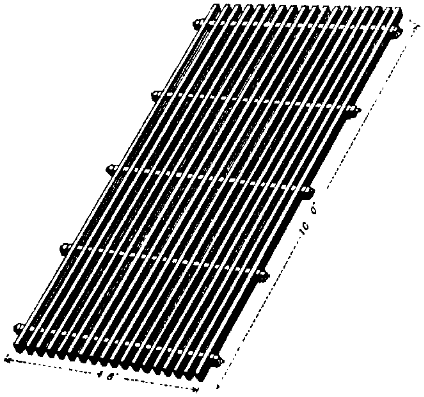

Fig. 4. Perspective view of Grizzley.

Grizzley. The grizzley shewn in Fig. 4 is largely used for the preliminary screening of ores, as the first stage of dressing. It consists of iron bars placed on edge, and held at the required distance apart by bolts passing through them, with washers threaded upon them alternately with the bars. Such washers can be made either from short lengths of tube or by bending pieces of flat iron. The grizzley bars are sometimes made of flat iron, 2 to 3 inches deep, ½ to 1 inch thick, spaced ½ to 3 inches apart. It is better that the bars should taper downwards as in Fig. 4, and as shewn in cross-section Fig. 5, so that any pieces of mineral that can enter the slots, are able to drop through, and thus avoid choking the bars. Old T-headed rails, placed heads downwards, make very good rough grizzley bars. With all these bars however there is a chance that a small flat piece of mineral may ride down on top of the bars instead of dropping through the spaces between them. To avoid this objection the top of the bar has been made ridge-shaped as in Fig. 6, this being the shape adopted at the Bascoup Colliery in Belgium, whilst Fig. 7 shews patterns adopted for the same reason in Pennsylvania. Grizzleys are usually 6 to 12 feet long and 3 to 6 feet wide; 10 feet long and 4 feet wide are very usual dimensions. The angle which the bars make with the horizontal is usually between 40° and 60°; it should be such as to allow the mineral to slide down smoothly and quietly, and so slowly as to give ample time for screening to take place.

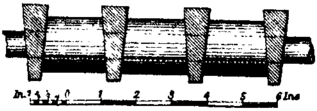

Fig. 5. Section of Grizzley bars.

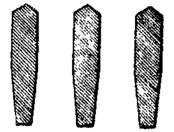

Fig. 6. Section of Grizzley bars.

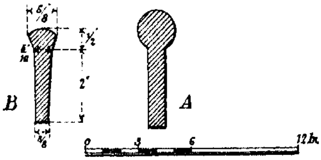

Fig. 7. Sections of Grizzley bars.

The grizzley should always be so situated that the mine cars can be brought in at such a level as to be tipped on to the head of the grizzley, whilst the oversize falls from the foot into other cars or else passes to the next process it has to undergo. The undersize, that passes through, is either collected in bins or else drops into a conveyor that removes it as fast as it drops down.

Grizzleys are sometimes also arranged for washing dirty mineral, a stream of water playing onto the latter as it slides down the bars.

Usually the spacing of the grizzley-bars is fixed definitely at the time of construction, but it sometimes, though rarely, happens that the width of slot is required to be adjustable within certain limits. This is best attained by making the distance pieces between the bars of a number of comparatively thin horseshoe-shaped washers, one or more of which can be taken out from or inserted into each space as required. Additional bars can also then be put in or removed so as to keep the total width of the grizzley always the same. The same object is sometimes attained by a complicated arrangement of long right and left handed screws passing through all the bars and by various other devices, which present the advantage that any change required can be made more rapidly than by the first method; they are however generally looked upon as too complex and delicate for such rough work as screening coarse mineral.

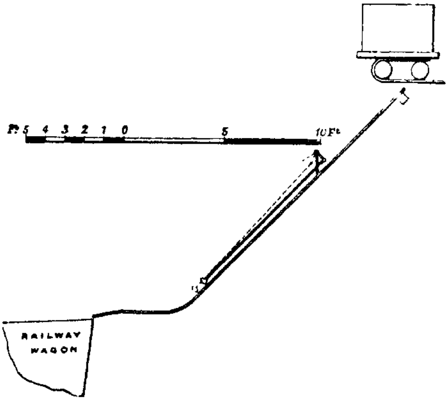

Fig. 8. General arrangement of Fixed Colliery screen.

Coal Screens. Fixed screens were at one time largely used for screening coals, but, for the reasons given below, their use has been almost entirely discontinued. They consist of bars of flat iron usually about ½ inch by 1½ inches spaced ½ inch to 1 inch apart, set at an angle of about 45°; the screens are usually 14 feet to 18 feet long and 3 feet 6 inches wide. About ⅓ way down a flap door (*B*, Fig. 9) is placed turning about its upper horizontal edge; this is usually closed, so that when a tub is tipped the coals can only slide as far as the door. When the sorter at the foot of the screen is ready for the coals he lifts the door by pulling the handle and lets the coals slide quietly down the screen; the object of this arrangement, known in various districts as the "Kepper" or "Keeper," is to let the coals down gradually so as to minimise breakage and also to keep the coals of each tub separate till they have passed the sorters. The general arrangement is shewn in one of its usual forms in Fig. 8, and the detail on a larger scale in Fig. 9[1].

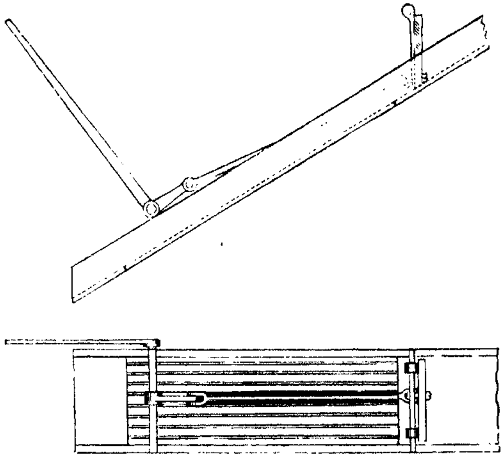

Fig. 9. Fixed screen with "Keeper."

Mention may be made here of a type of screen which though really fixed, resembles moving screens in its results; this is the McDermott sizer, which consists of a fixed inclined screen of small mesh, over which material is caused to travel by the pulsations of the water in a box in which the screen is fixed, these pulsations being caused as in the ordinary jig (see page 257). It is said to do good work on fine sizes.

2. Moving Screens. The objections to fixed screens are that they rapidly become choked up or "dumb" in screening wet material; that small pieces may ride down upon larger ones and the sizing thus be very imperfect; that they require a great deal of head-room; that they need attention and more or less hand labour; and finally that they break up the mineral a great deal, this last objection applying more especially to the screening of such a material as coal, in which the large lumps or "round" coal are in many cases worth very much more than the small. Accordingly it will be found that a very large proportion of moving screens have been devised purposely for the sizing of coal, the main object in view being to accomplish a thorough sizing with as little breakage as possible of the coal. Two or more, sometimes as many as half a dozen, sizes may be made simultaneously, the requisite number of screens being arranged either vertically or horizontally in their proper order, and either worked independently of each other, or else all simultaneously by the same mechanism. There are obviously two ways of arranging such multiple screens: either those with the largest apertures may come first, when all that refuses to pass through the first screen or its over-size forms the coarsest size, whilst the material that passes through, or its under-size, goes to the next screen, and so on. The refuse or over-size from each screen thus forms the sized material, whilst the screenings that pass through, or the under-size, need further sizing. In the other method the finest screen comes first, and then the under-size from each sieve form the sized material, whilst the over-size has to be sized further. The former system lends itself the more readily to the nesting of flat screens of the type now under consideration, and is moreover free from the serious objection that if the finest screen comes first it is apt to be worn out unduly by the coarse material passing over it; coarse screens on the other hand being made of stouter material can better resist the wear due to large rough fragments and to the larger amount of material that passes over them if the first method is used. Hence we shall find that the former method is practically always employed except in the cases in which a series of screening surfaces are set one following the other in the same plane, when the first must necessarily be the finest. Such screens will here be called compound screens, whilst those arranged on the first plan, one over the other, will for the sake of distinction be spoken of as complex screens.

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[1] *Trans. Min. Inst. Scot.* Vol. xi. p. 183.

2 (*a*). **Screens, the screening bars of which alone move** (independent motion of bars). These screens are mostly bar screens or screens of some similar type, for the obvious reason that quite independent motion can only be attained when bars are used; with punched plates or woven

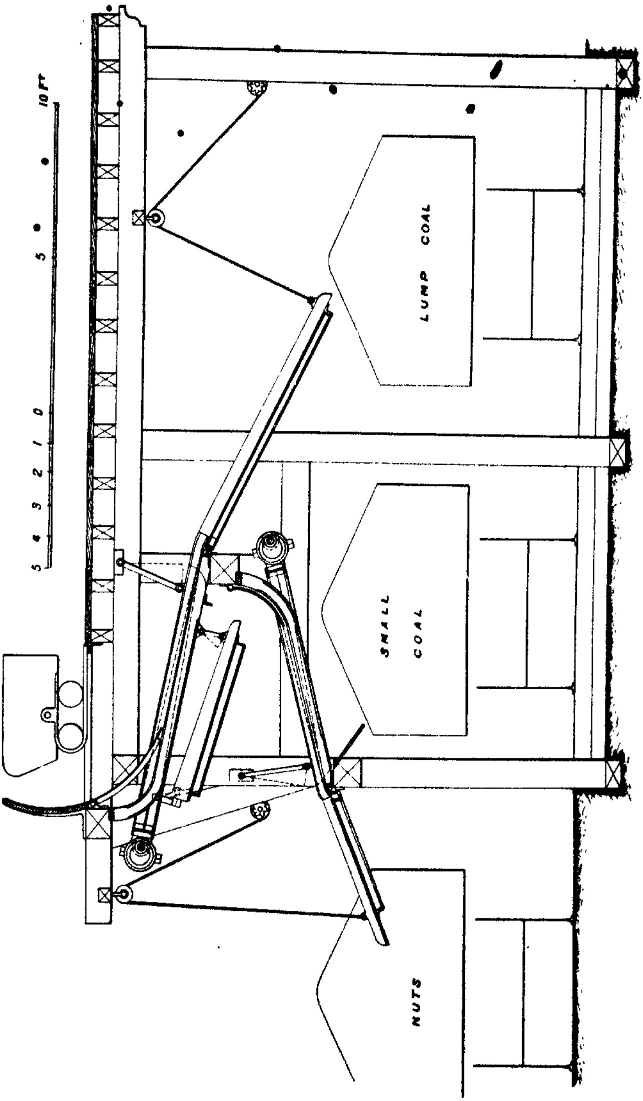

Fig. 10. Old Briart screen

wire the entire screening surface, or at least a considerable section of it, must of course move as one piece. The objection already stated to bar screens, namely that they allow large pieces to pass through if flat, has to some extent militated against the more extended use of some of these screens.

Briart Screen. This screen was designed by M. Alphonse Briart and employed in 1872 at the Bascoup and Mariemont Collieries. The original form is shewn in Figs. 10 and 10a[1]. It consists of bars alternately fixed and moving, all being in one plane when the latter are at rest in one particular position. The alternate moving bars are secured in a frame the lower end of which is hung by means of swinging links, whilst the upper end is connected by rods to a pair of eccentrics keyed in parallel positions on a driving shaft. During one half of the revolution of this shaft, the moving bars are above the plane of the fixed bars, and below that plane during the other half. The direction of revolution of the shaft is such that the eccentric is moving in the direction from the top to the bottom of the bars during the first half; hence when coals are thrown upon the upper end of the screen,

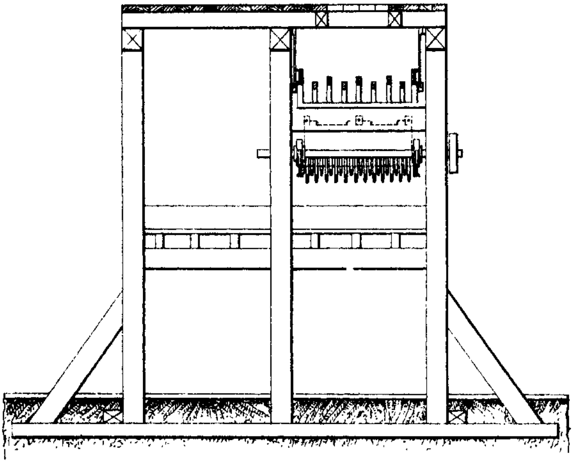

Fig. 10a. Old Briart screen.

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[1] *Publ. de la Soc. du Hainaut,* "Note sur un système de Triage Mécanique," by Alph. Briart, Series II. Vol. IV. 1873, p. 57.

those resting upon the moving bars are carried forward towards th lower end during this half of the revolution of the shaft, and are at res during the other half, supported by the fixed bars. At the above name collieries pairs of screens are arranged one above the other, the bar of the upper one being about 6 inches and of the lower one 1·5 inches apart; the capacity of this plant was about 60 tons per hour and its power consumption 2 H.P., the shaft making 35 revolutions per minute. It was found that this screen made fully 5 per cent. less smalls than the

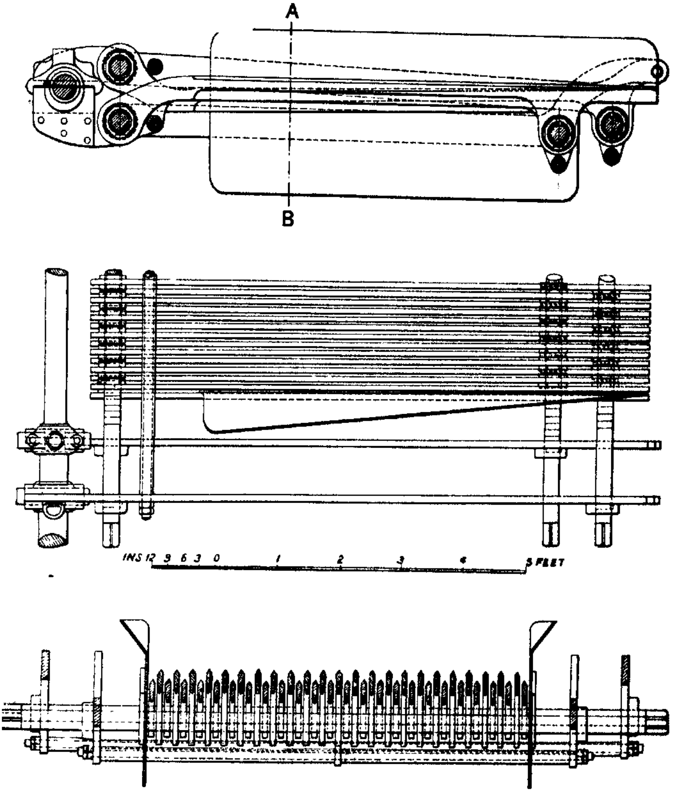

Fig. 11. Newer form of Briart screen.

old type with fixed bars. In some careful trials it was found that a single screen could work up to a maximum of 80 to 100 tons per hour, taking 1·626 I.H.P., of which 35 per cent. was absorbed by the engine, 38 per cent. by the screen running empty, and 27 per cent. in moving the coal.

This screen was soon improved by making both sets of bars movable; each is attached to a suspended frame and each is actuated by cranks or eccentrics set at 180° to each other, the direction of revolution being such that it is always the set of bars that is above the other that is being moved from the top towards the bottom of the screen. The coal is thus carried forward during one half of the revolution by one set of bars and by the other set during the second half. The capacity of the screen is thus greatly increased, whilst the coal does not undergo any rougher treatment. Such a screen is shewn in Fig. 11.

The objection that flat pieces may fall down between the bars has been overcome by an arrangement adopted in several Westphalian Collieries. Here the bars consist of channel iron bottom uppermost, the bottom web being punched with round holes through which the coal is screened, whilst the bars themselves are only 0·2 inch apart; the bars are 4 inches wide and are punched with 3 inch holes, as shewn in Fig. 12[1]. A complete screen, as built on this principle by the Humboldt Engineering Company, is shewn in Fig. 13.

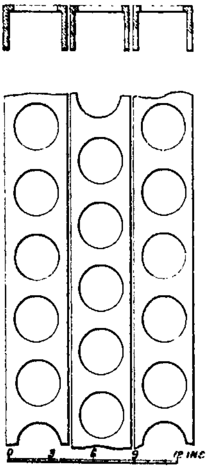

Fig. 12. Section of bars from Briart screen.

Coxe screen[2]. In this screen, Figs. 14 and 15, designed by Eckley B. Coxe, the bars have the peculiar curved shape shewn in Fig. 14. The upper ends of the bars rest upon iron rollers, the latter being carried on horizontal steel plates, so that the bars have perfect freedom of movement in a horizontal direction. The lower ends

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[1] *Bull. de la Soc. de l'Ind. Minérale*, "Voyage en Westphalie," by Mm. Mathet, De Gournay and Suisse, Series III. Vol. I. 1887, p. 65.

[2] *Trans. Amer. Inst. Min. Eng.* Vol. XIX. p. 402.

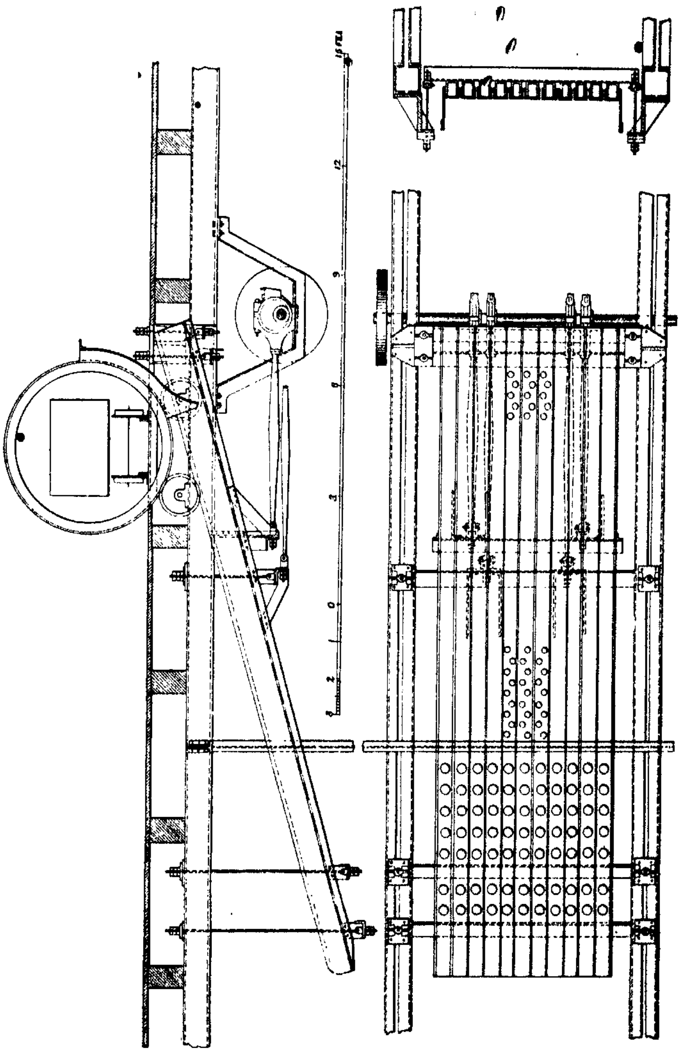

Fig. 13. Humboldt type of Briart screen.

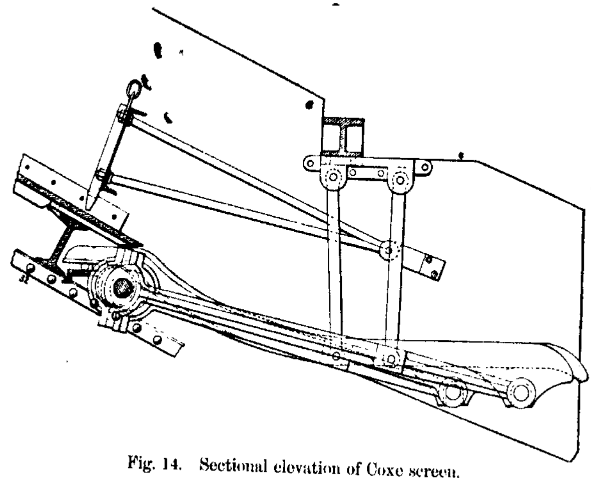

Fig. 14. Sectional elevation of Coxe screen.

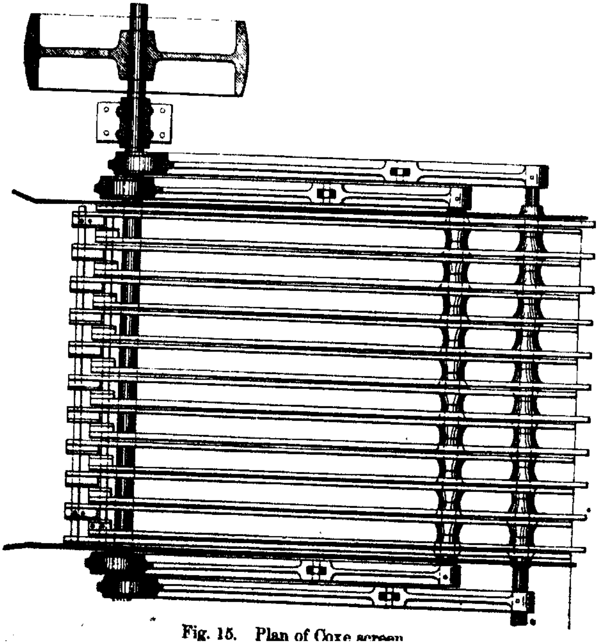

Fig. 15. Plan of Coxe screen

of the bars are provided with semicircular notches, which rest on a pair of transverse shafts situated in the same plane, but one about 18 inches in front of the other. The bars rest alternately on the front and on the back shaft, each set partaking of course of the motion of the shaft upon which it rests. The shafts are connected by means of eccentric rods, each to two pairs of eccentrics keyed on one driving shaft, at angles of 180° from each other. The throw of the eccentrics is 3 inches. Each eccentric rod is suspended by a swinging link attached to a pin passing through the rod, the position of the pins being such that their distance from the centre of the carrying shafts is one half of the distance from the pins to the centre of the eccentric discs. The links being comparatively long, when the eccentric shaft is revolved, the pins which connect the links and eccentric rods move in an arc of a circle so large that it may be looked upon as practically a straight line, so that these pins travel backwards and forwards practically 3 inches. As the supporting shafts are only half as far from these pins as the eccentrics, they have only one half of the vertical motion of the latter, hence the shafts carrying the ends of the screen bars describe what is practically an ellipse, of which the horizontal major axis is 3 inches and the minor axis 1½ inches. A diagram of the motion is shewn in Fig. 16, where *A* is the point of suspension of the link and *B* the centre of the eccentric shaft, the

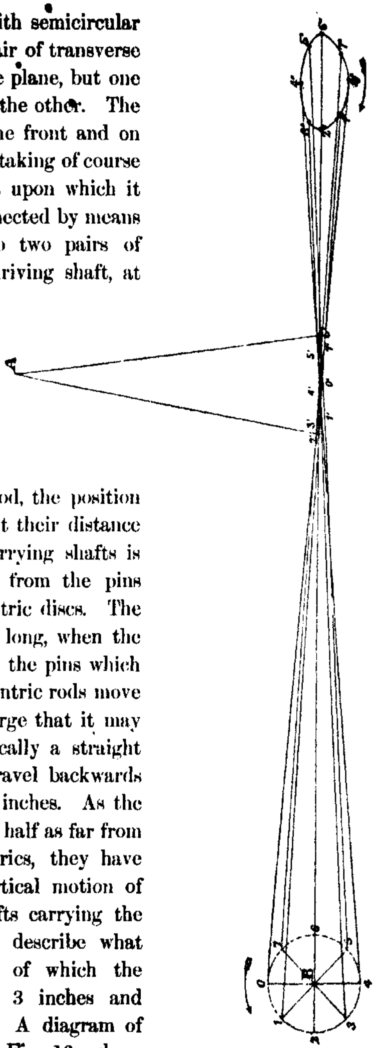

Fig. 16. Diagram of motion of Coxe screen.

successive positions of the bars being shewn at 0, 1, ... 7, corresponding to the similarly numbered positions of the eccentric. It is evident that the path described by the bars is nearly a true ellipse. As in the Briart screen the set of bars which is above the other is always moving forwards, the back stroke taking place whilst it is below the other set. Hence the anthracite (for which this screen is used) is carried forwards by 6 inches in each complete revolution of the driving shaft, whilst the rise does not exceed ¾ inch, so that in this screen tender coal can be screened without being thrown up and down at all violently as may be the case with the Briart screen when worked fast. The Coxe screen shewn in the figure, about 9 ft. long × 5 ft. wide, lying at an angle of about 6°, has a capacity of about 120 tons per hour.

Chambers screen. This screen is the invention of Mr Chambers, and is in operation at the Denaby and Cadeby Main Collieries and others. It is shewn in plan and elevation in Fig. 17 and in perspective in Fig. 18. In this screen, as in the improved Briart, the bars are not solid, allowing the coal to drop through the interspaces between them, but they lie so close to each other that there is practically no screening space between them, the smalls dropping through apertures in the bars themselves. These bars are of cast steel and have the form of inverted channel irons with square apertures in the flat upper surface. All the alternate bars are coupled together, so as to form two separate sets. The bars are set in motion by two pairs of eccentrics keyed at 180° to each other. One pair of eccentrics drives front and back rocking bars, *A* and *B*, having the shape of inverted T's, from which project pins that fit into corresponding recesses in the screen bars and give alternate sets an up and down motion, whilst the other pair of eccentrics drives another rocking link, *C*, the two ends of which move alternate sets of bars back and forwards. The result of this arrangement is a back and forwards motion of each set of bars combined with an up and down motion, so arranged that the forward motion takes place whilst the bar is moving upwards, so that, as in the previous examples, the coals are moved slowly along the screen.

This screen is unusually free from injurious vibration, is relatively silent in operation, is simple and easily repaired, and the bars can be taken out and others with different sized perforations substituted rapidly and readily; any size between 1 inch and 6 inches square

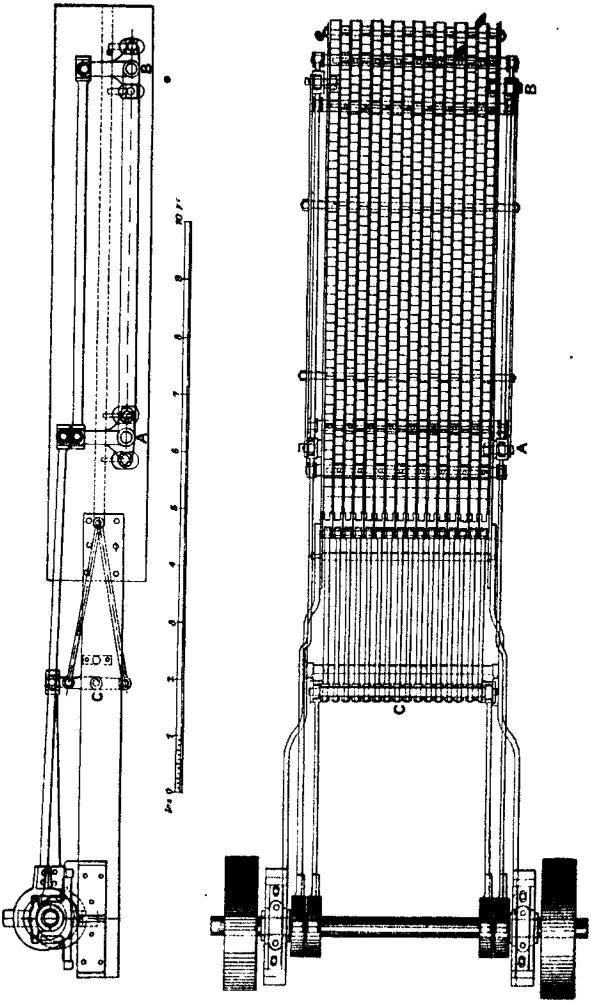

Fig. 17. Plan and elevation of Chambers screen.

may thus be made. The screen bars are usually 12 feet long and the screens themselves are usually from 3 ft. 6 in. to 6 ft. wide. A screen 4 ft. 6 in. wide will treat about 120 tons per hour and requires about 3 H.P. to drive it; its cost is about £125.

Borgmann and Emde screen[1]. This differs a good deal from those previously considered. The type shewn in Fig. 19, manufactured by Messrs Schüchterman and Kremer, consists of a number of fixed longitudinal bars, rectangular in section, whilst the transverse bars of round iron are caused to revolve. The longitudinal bars consist of flat iron set on edge, in the upper edges of which semicircular notches are cut. These support the transverse round bars, each of which terminates in a sprocket wheel, so that all can be simultaneously driven in the same direction by means of an endless chain into the links of which all the wheels gear. Instead of this driving arrangement an outside lay shaft parallel to the length of the screen may be used, driving the bars by means of bevil gearing. The coal thrown onto the screen is carried forward frictionally by the rotation of the bars. This screen is used in Bochum; it will work when horizontal, hence needs a minimum of headroom, is very free from jar, and does not break the coals at all; it also has the advantage of having square apertures, but on the other hand it is decidedly heavy. A usual construction is with round bars 1¼ inches in diameter, set 3 inches apart in the clear. The flat bars are also 3 inches apart in the clear; the screen is 7 feet long with 18 round bars making 50 to 60 revolutions per minute.

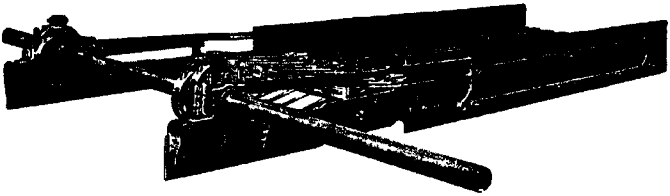

Fig. 18. Perspective view of Chambers screen.

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[1] *Zeitschr. f. B. H. u. S. Wesen im Preus. Staat.* Vol. xxxv. p. 264.

A modification of this screen has been introduced at Bethune[1] as shewn in Fig. 20. It differs from the last in that the longitudinal flat iron bars are given a small up and down motion, imparted to their upper ends by an eccentric, whilst the lower ends pivot on a fixed bar. The round transverse bars and the flat longitudinal bars are respectively 3·1 inches apart, thus forming a square mesh of this size. The

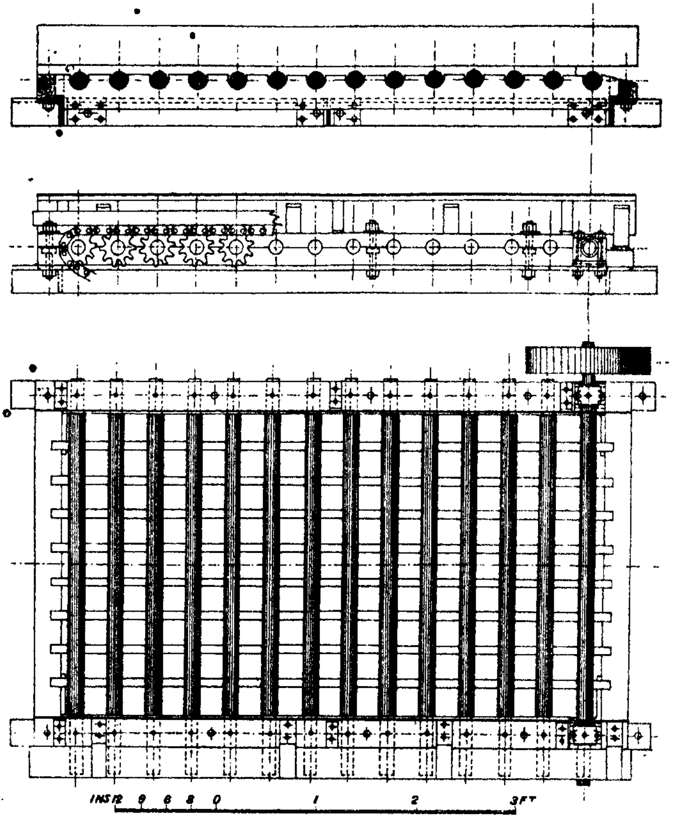

Fig. 19. Borgmann and Emde screen.

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[1] *Bull. Soc. Ind. Min.* Vol. xv. 1901, p. 401.

mechanism is somewhat complicated, but the results are said to be very satisfactory.

There are certain other forms of screen that ought strictly to be included here, namely screens of which only the screening surface moves, but which are not bar screens; such are for example certain forms of picking belts which are arranged to act

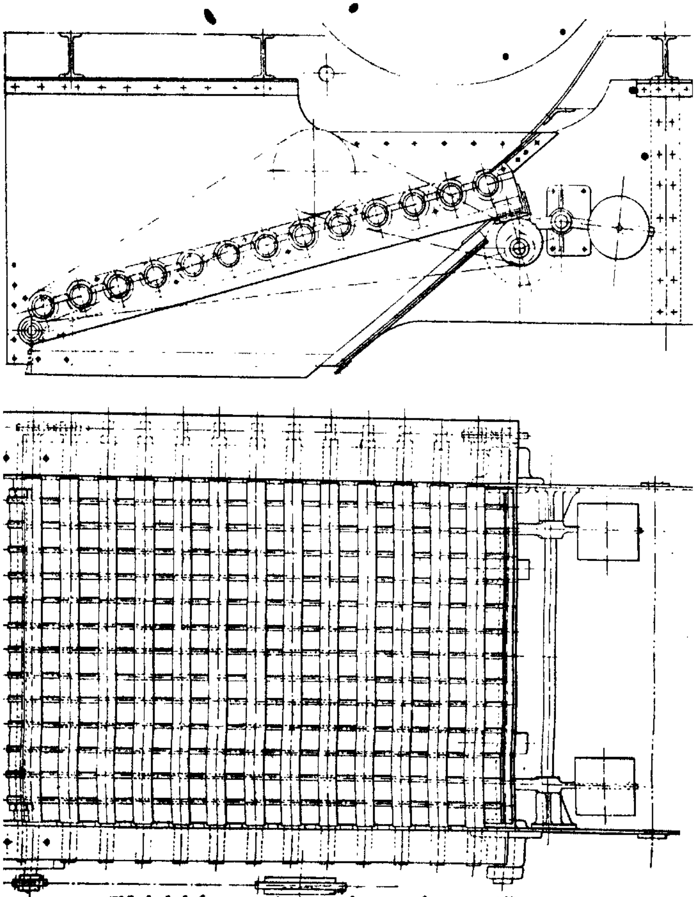
Fig. 20. Bethune screen.

consisting of endless belts of various types, with perforated surfaces. These will be considered under the head of picking belts (see p. 96). Again there is a form of vibrating screen, in which the screening surface, usually of wire gauze attached to a light frame vibrates inside what in other types is looked upon as the frame proper; this however is better classified as a modification of vibrating screens (see p. 38), rather than under this head, as the screening surface moves as a whole.

2 (*b*). **Screens in which the entire appliance moves.** In this class we shall find a comparatively small number of coarse screens, whilst secondary or fine screens belong to it almost exclusively; the reason, obviously enough, is that the previous class consists of bar screens which are only suitable to coarse sizing, whilst fine wire gauze and perforated plates, which are necessarily required for fine sizing, can be employed in most forms of the class next to be considered.

This class may be divided into three groups according as the motion imparted to the screen is

(α) Reciprocating,
(β) Gyrating,
(γ) Rotating.

(α) **Reciprocating screens** are often spoken of as Shaking or Vibrating screens. The motion may be either a "bump" or a true shake. In bumping screens, the shock at the end of each stroke is very effective in securing thorough screening, and, in most forms, also in moving the mineral forward, so that a very flat grade will suffice; at the same time it is apt to break up tender minerals and thus to make smalls. For this reason we shall find bumping screens almost confined to the sizing of ores, whilst vibrating screens are preferred for coal.

Bumping screens with vertical motion. In this arrangement the screen may be hinged at its lower end, the upper end being lifted by a cam and allowed to drop onto a massive wooden block or bumper. The screen should be so arranged that when the top end is in its lowest position the mineral to be screened cannot slide down it, but when lifted by the cam its *angle to the horizontal should just exceed the* angle of repose of the mineral. In this way the mineral will slide down a little way at each stroke, and the bump need only be severe enough to shake the pieces of mineral so as to prevent small pieces from riding on the larger ones.

It is more usual to hinge these screens at the upper end and to lift the lower by a cam, allowing it to drop sharply against a bumping piece. In this case the angle of the screen at no time exceeds the angle of repose and there is no tendency on the part of the mineral to slide down the screen. When the screen is moved, and the mineral with it, the latter has a certain amount of momentum imparted to it; when the screen is suddenly stopped by striking on the bumping piece, the mineral tends to continue its motion downwards, and, as the result, is moved down the inclined plane formed by the screen. Th rate at which the mineral is advanced at each stroke will be great the greater the inclination of the screen and the mass of the individu pieces of mineral, and the less the friction between the pieces of miner and the screen. These screens are used a good deal in Germany; a typical example is shewn in Fig. 21.

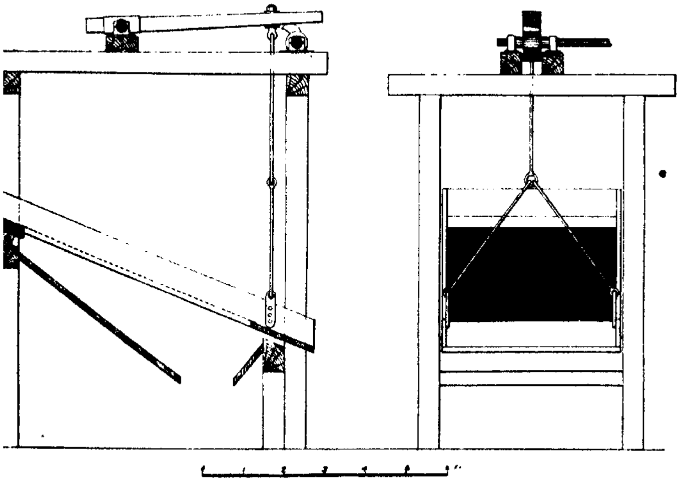

Fig. 21. Vertical bumping screen.

Bumping screens with horizontal motion may move in either a longitudinal or a transverse direction; the former is preferable because it aids the travel of the mineral along the screen, whilst it is, at the

Such screens are thrust slowly in the required direction by means of a cam and then fall back sharply again, the moment the point of the cam has cleared the tappet against which it presses; the slow movement upwards of the screen does not displace the mineral on it, but the latter is jerked forward owing to the momentum it acquires during the rapid fall back of the screen; this effect may be accentuated by causing the screen to strike against a bumping block. In many screens, however, the rapid return of the screen to its original position is sufficient without any definite bump. A screen of the latter type is illustrated in Fig. 22. The cam shaft is vertical and the cam acts upon a tappet which slides along the tail rod and can be set at any desired point so as to enable the amount of throw to be varied as required. These screens are often made double, thus making three sizes, but each of the screens is suspended independently and moved by its own single-toothed cam, these cams being set at 180° to each other so as to balance the strain on the driving engine.

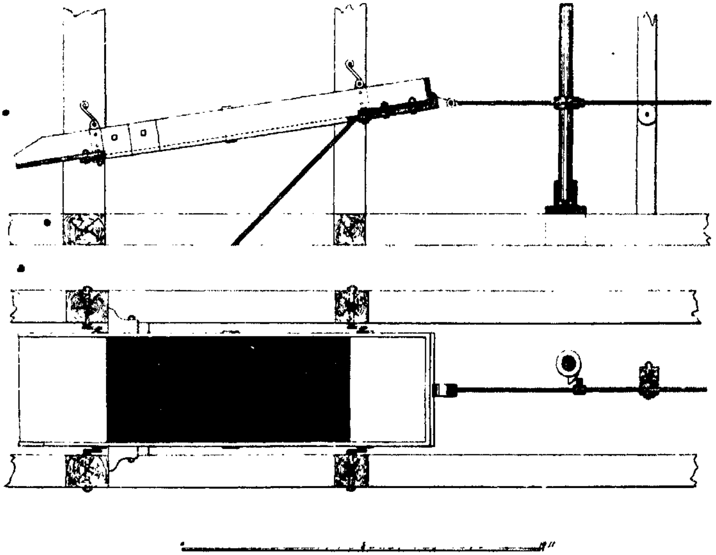

Fig. 22. Horizontal bumping screen.

Screens of the last two kinds are of about equal efficiency; their lengths are usually 3 to 6 feet, with a width corresponding to the amount of mineral to be treated; their inclination to the horizontal varies generally between 10° and 20°; the number of blows per minute is usually between 100 and 200, the length of stroke varying from 1 to 4 inches. An ordinary screen about 6 ft. × 3 ft. will treat 10 to 30 tons of mineral per hour and will absorb 1 to 2 H.P. Both of these screens are better suited to ores than to coal as they tend to break up the mineral a good deal.

Shaking or vibrating screens, often called **Jigging Screens**, are very largely used both for coal and for other minerals; they are usually set in motion by means of a simple eccentric as shewn in Fig. 23, which represents a screen built by Messrs Head, Wrightson & Co., Ltd. Screens of this type are arranged either as simple, as compound, or as complex screens. The screening surface may be of any kind, gauze, locket work, bars, and perforated plates with apertures of various shapes all being used; bars are not however much in favour, whilst the lightness of gauze screens leads to a preference for this material. The entire screen with sheet iron sides and hopper beneath may be moved as shewn in Fig. 23, being in that case suspended by means of links or rods. The suspension usually allows the angle of the screen to be altered when desired. Another plan, shewn in Fig. 24, which represents a screen built by Messrs Coulson & Co., is for the sides of the screen and the hopper to be fixed, whilst the screen vibrates supported merely on a light frame, hung from short arms; the latter are usually pivoted on the upper edge of the fixed sides and are

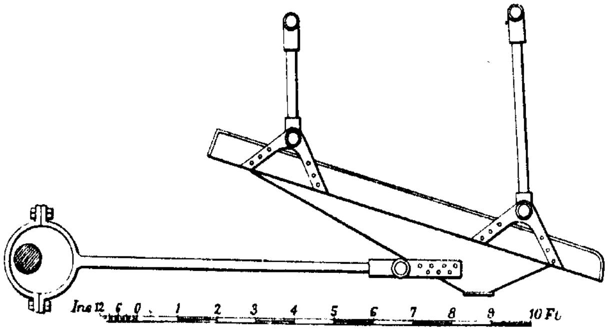

Fig. 23. Plain jigging screen.

outside the latter, which are provided with curved slots in which the pins; that connect the screens to the arms, can move freely. This system is somewhat more complicated and does not as a rule allow the angle of the screens to be changed; it may also make a trifle more smalls than the former method, but presents the advantage of compactness, and of a much lighter moving load.

The angle of jigging screens as used for coals is usually between 10° and 18° to the horizontal, 14° being an average inclination; the length of stroke is usually between 5 and 9 inches, and the number between 70 and 120; as a general rule these last two figures vary inversely, and the number of strokes per minute, multiplied by the length of strokes in inches, should give a product that does not vary much from 600, or perhaps a little less for very short strokes and rather more for long ones. These screens are usually 4 to 5 feet wide and 8 to 10 feet long; their capacity is between 40 and 100 tons of coal per hour according to circumstances, and the power required to work a screen is usually from 2 to 3 I.H.P. When two or more screens working independently, i.e. not "nested," are used, their eccentrics are best all keyed on one shaft and so arranged as to balance each other and to keep the strain on the driving power as

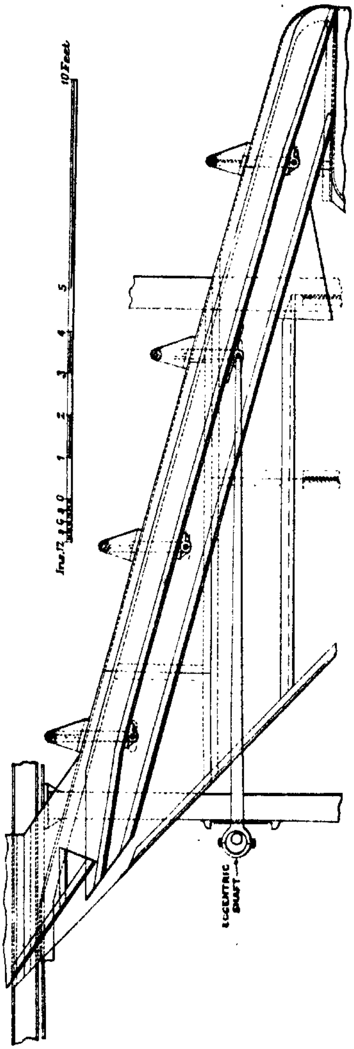

Fig. 24. Jigging screen with fixed sides.

uniform as possible. These screens are noisy and tend to rack the structure in which they work a great deal, but are simple, effective, easily kept in order or repaired, and are hence much used. They also admit of the screening surface being readily changed when the sizes have to be varied, or of the screens being "plated" or rendered "dumb" when they are required to discharge unscreened coal or "run of mine."

The **Klein Screen** made by the Humboldt Engineering Company is shewn in perspective in Fig. 25, and in plan, elevation and section in Fig. 26[1]. As shewn, the screen is carried upon a swinging frame suspended by means of links, the frame being actuated by means of a crank shaft close to the back of the screen. The lift is thus a maximum at the back end, and the upstroke is much quicker than the downstroke, whilst the whole screen receives a rapid to and fro movement, as shewn in the diagram Fig. 27, the upward motion being about 2½ times as rapid as the downward motion. The coal is thus thrown forward rapidly and uniformly by this motion even though the screening surface be absolutely horizontal; a very high capacity is claimed for this screen. A screen 13 feet 6 inches long and 4 feet 6 inches wide

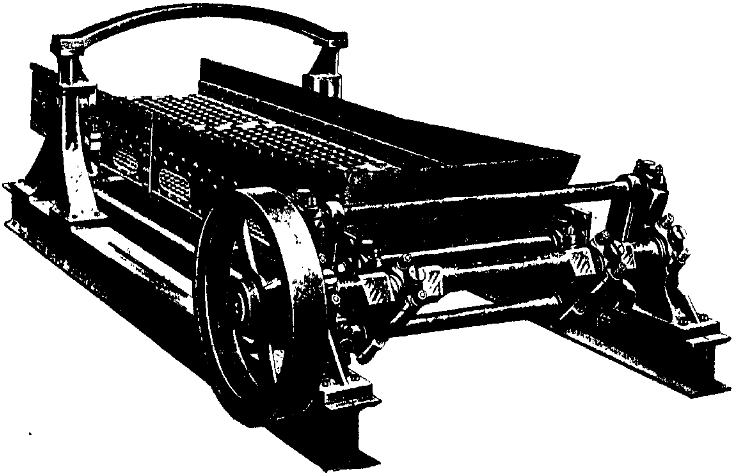

Fig. 25. Perspective view of Klein screen.

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[1] *Trans. Fed. Inst. M. E.* "Anthracite coal-breaking and sizing plant at Glyncastle colliery," by W. D. Wright, Vol. XII. 1896–97, p. 238.

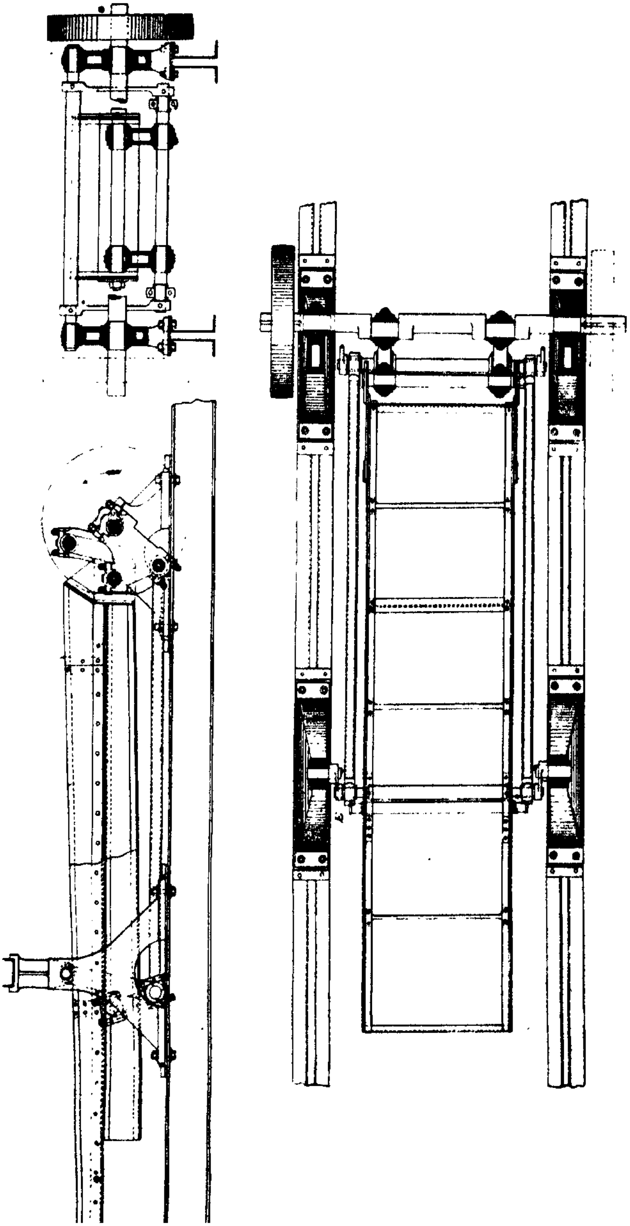


Fig. 26. Elevation, plan and section of Klein screen.

with 2 inch mesh, making 130 2½ inch strokes per minute, will screen close upon 150 tons of coal per hour.

Somewhat similar to the Klein screen, but rather simpler in construction, are the **Laue Screen**, made by Messrs Schüchtermann and Kremer, shewn in Figs. 28 and 28^a, and **Durnford and Wormald's Screen**, made by Messrs Head, Wrightson and Co., Ltd., shewn in Fig. 29, both being arranged as complex screens, with two screening surfaces, one above the other as indicated.

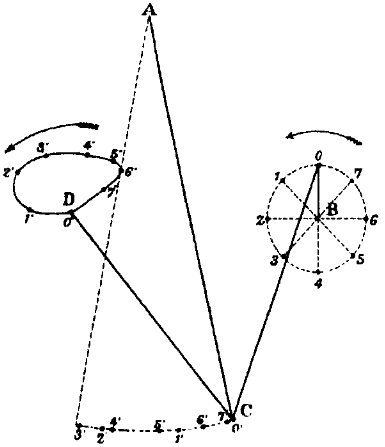


Fig. 27. Diagram of motion of Klein screen.

Another form of screen, made up of a number of short transverse plates, each of which receives a rocking motion from one pair of eccentrics, is shewn in Fig. 30. This is known as the "**Anti-breakage**" **Screen**, and is also made by Messrs Head, Wrightson and Co., Ltd. Each plate in turn rocks to and fro, and thus moves the coal along from the head to the foot of the screen, which must be inclined at a moderate angle; the motion is a fairly smooth one, but there are many wearing parts.

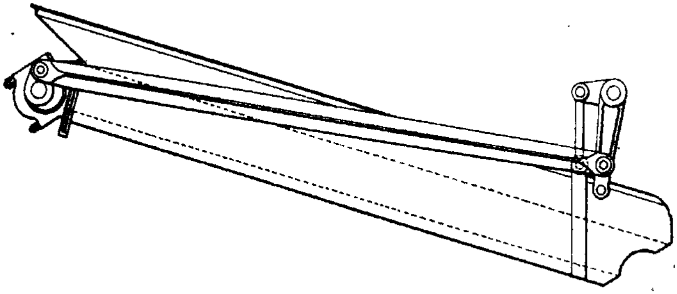
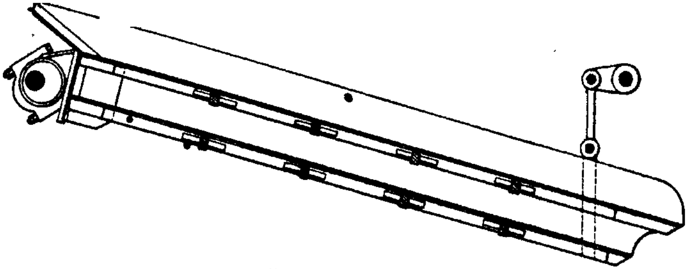
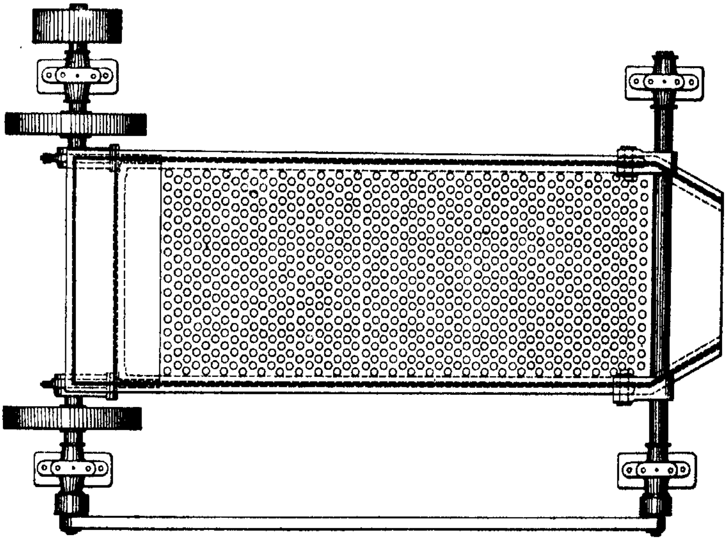


Fig. 28. Elevation of Laue screen.

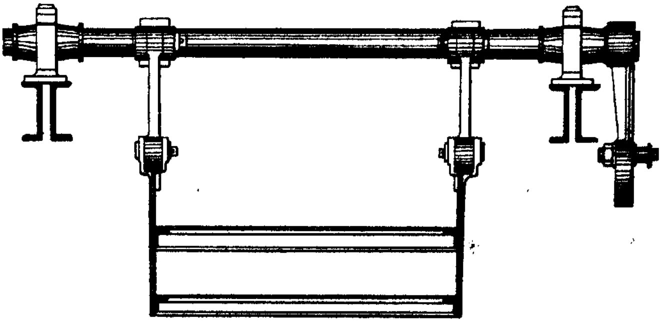
A special form of screen known as the **Impact Screen** has



Sectional elevation.

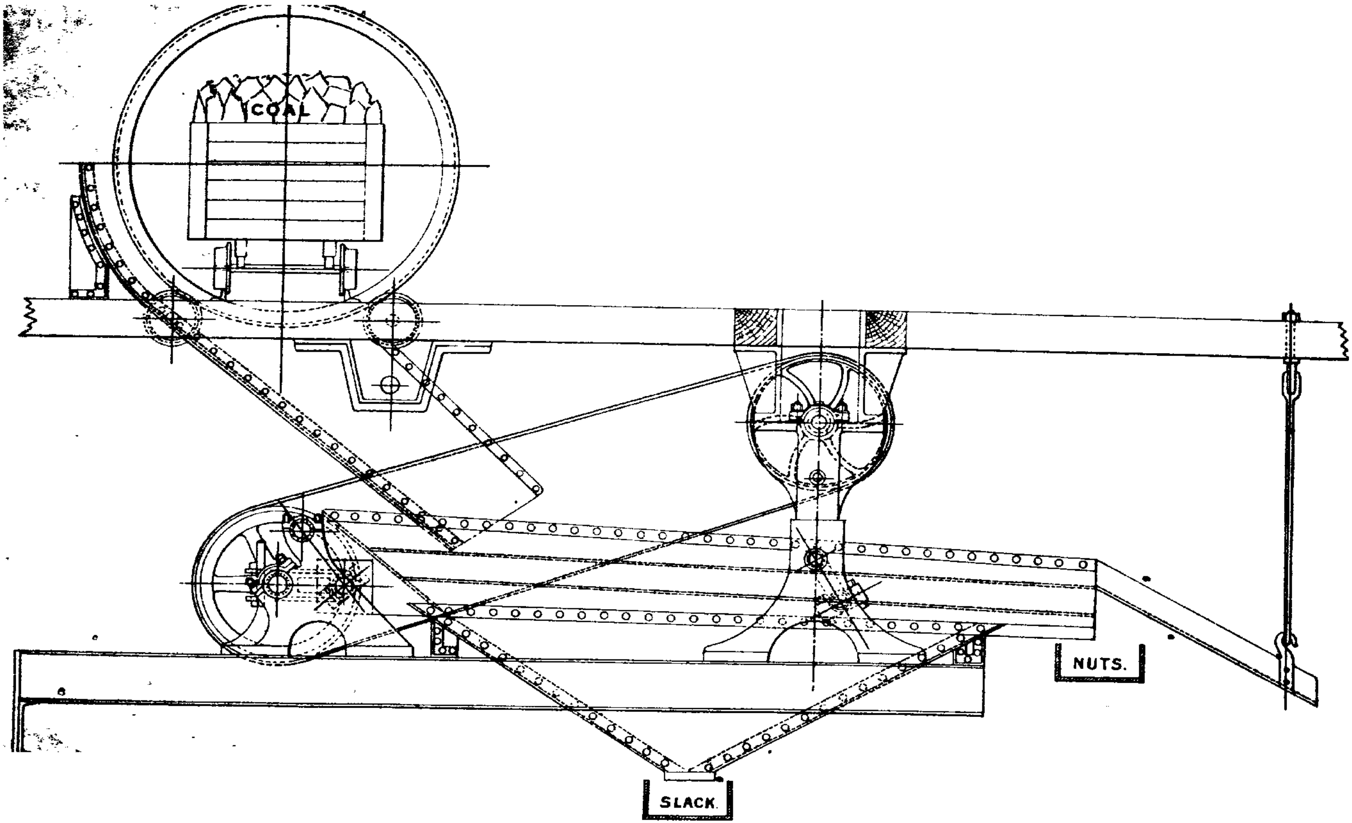


Plan.



Transverse section.

Fig. 28ᵃ. Longitudinal section, plan, and transverse section of Laue screen.


COAL
NUTS.
SLACK.

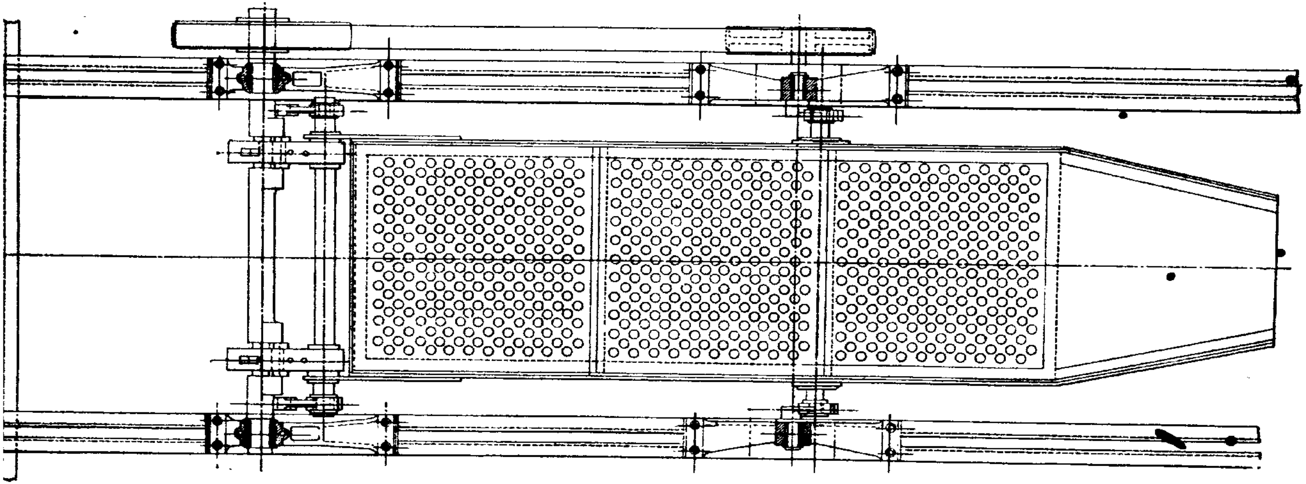


Fig. 29. Elevation and plan of Durnford and Wormald's screen.

lately been introduced by the Wilfley Ore Concentrator Syndicate, Ltd., which consists of a screen usually of fine mesh and set at a rather steep angle, which receives downward jerks from a cam at the rate of 600 per minute, the screen being thrown upwards again by a spring. This screen is arranged to work either dry or wet; the makers claim that one screen 3 feet by 3 feet will do the work of two ordinary "trommels."

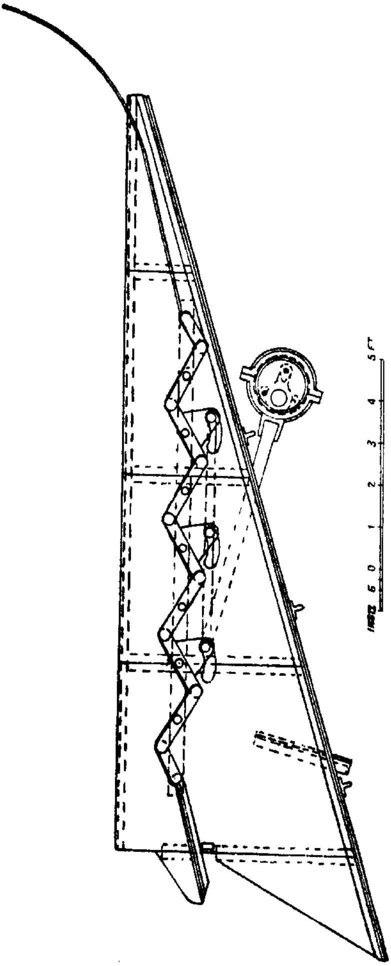


Fig. 30. "Anti-breakage" screen.

A special type of reciprocating screen has been introduced of late years, more particularly for screening coal, in which the oversize is carried along on the screen, often for considerable distances; these screens may hence be distinguished as "conveyor" screens.

The **Zimmer Conveying Screen**, Fig. 31, consists of a shallow trough with suitably perforated bottom, supported on short pivoted legs made of elastic wood, and thus acting in part as springs; it is worked by a crank shaft, the connecting rod being coupled to the trough by a spring attachment so as to jerk it forward. The Ferrari screen is practically identical with this form.

Marr's Excelsior Conveyor, Fig. 32, is also a trough, suspended from short arms, in which the advance of the coal is brought about by a

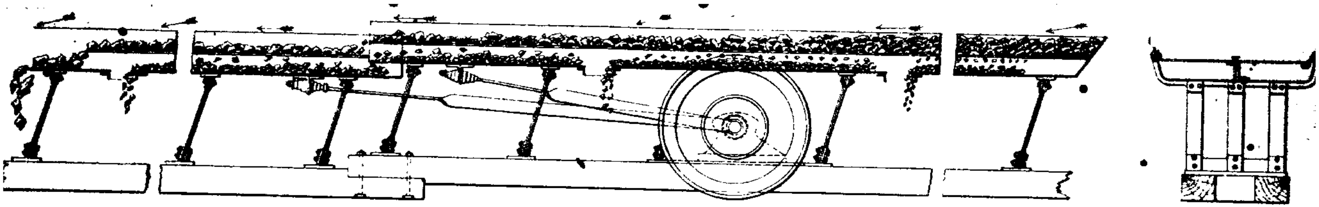
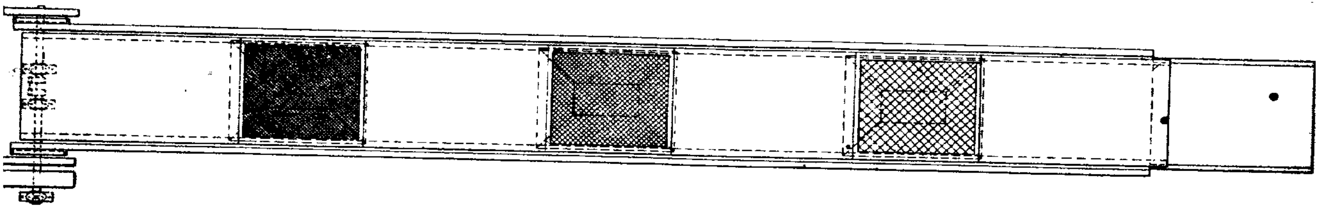
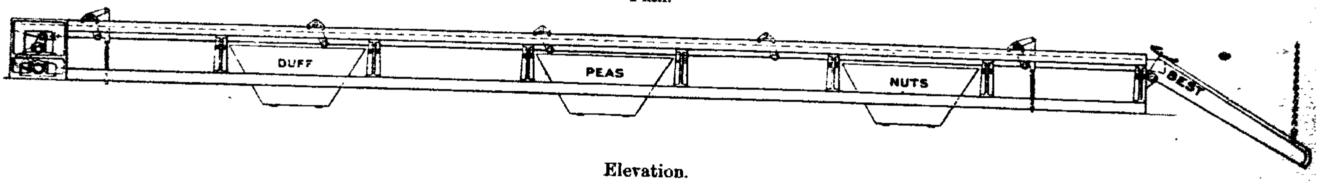


Fig. 31. Zimmer conveying screen in elevation and cross-section.



Plan.



Elevation.

Fig. 32. Marr Excelsior screen in plan and elevation.

special composite crank, as shewn in Fig. 33, on a larger scale, which gives a quick forward motion and a slow return, thus moving the coal along. Owing to the shortness of the arm there is a noticeable vertical motion as well as a horizontal one.

The **Marcus Screen** is somewhat similar in design to the last-named; the narrow screening trough is carried upon wheels or rollers, and receives a motion forwards which is slow at first and gradually accelerated for about three-fourths of the stroke, when it slows down again. The mineral is thus given sufficient momentum to enable it to overcome the friction of the surface upon which it rests and thus to move forwards; during the slower backstroke it is not thrown back again, and is therefore advanced over the screen. The action is smooth, owing to the slowing down at the end of the stroke, and the appliance is quite efficient, though running only at 60 to 80 strokes per minute. Marcus screens have recently been erected capable of dealing with from 35 to 150 tons of coal per hour.

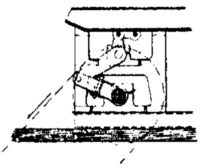


Fig. 33.
Driving mechanism of the Marr Excelsior screen.

The **Kreiss Screen** is very like the Zimmer. In France[1] a Kreiss screen has been used for making 5 sizes of coal, namely, 0·4 inch, 0·7 inch, 1·4 inch, 1·8 inch, and 2·4 inch. The entire screen is 21 feet 6 inches long and 24 inches wide, and the screens are arranged successively, the smallest mesh coming first. The entire screen has a slope of 3°, and is carried on four legs 13·4 inches long, which vibrate through an arc of 6°, namely from 68° to 74°; it is worked by a crank having a throw of 6·9 inches at 320 strokes per minute. The entire screen weighs 16 cwt., and the momentum due to its movement is absorbed by 3 nests of spiral springs placed at the upper end and 4 nests along each side (or 11 groups of springs in all) so arranged as to take up the movement either upwards or downwards. At the head of the screen is a Kreiss conveyor, attached to the screen and working with it, 13 feet in length, which brings the undersize from a screen of 3·15 inches mesh. This arrangement can handle 20 tons of coal per hour.

(β) **Gyrating screens.** This class comprises screens that have a motion of revolution round a vertical axis, the path of revolution being either elliptical or circular. In the former case the nature of the movement tends to approximate to that of the last class, the straight line being one of the ultimate figures to which the ellipse may be reduced; the other limit is the circle, and accordingly we find some gyrating screens, particles placed on which tend to move in a circular path, and others in which they move in ellipses of varying eccentricity.

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[1] *Comptes Rend. Soc. Ind. Min.* January, 1907, p. 21.

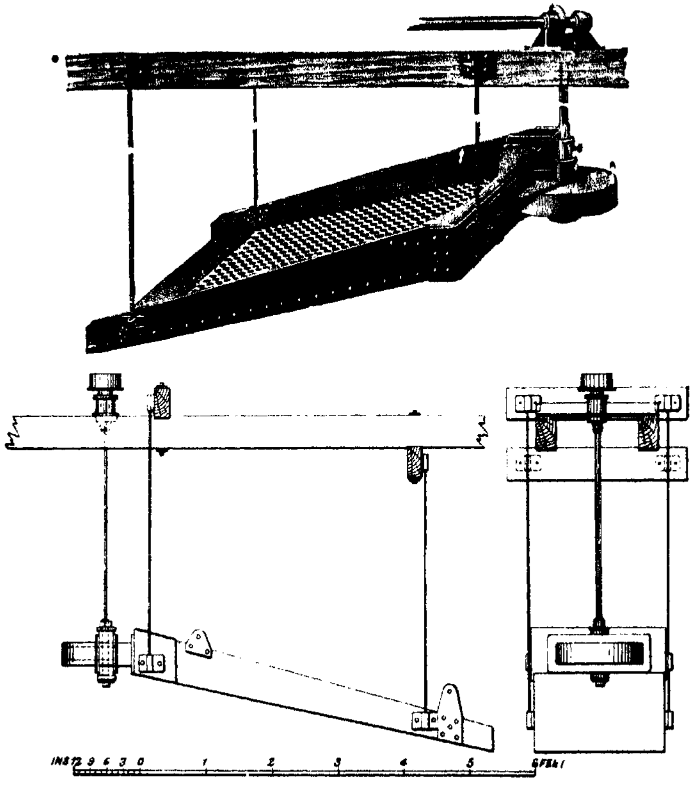

Fig. 34. Small vibromotor screen.

The motion of a gyrating screen resembles closely that given to a sieve in hand-riddling and may be looked upon as the most perfect type of screening motion.

Vibromotor screens. The vibromotor is a device invented by Mr W. Beaumont[1], and it has been applied to the screening of minerals by the Hardy Patent Pick Co., Ltd. The vibromotor *consists of* a vertical flexible or jointed shaft which is secured eccentrically to a heavy horizontal block or disc. When the shaft is rotated the point of insertion of the shaft in the weight describes a circle the diameter of which depends upon the velocity of rotation and on the weight of the moving parts. If the lower end of the shaft is attached to a screen freely suspended, e.g. by ropes, the screen will be gyrated with it, and owing

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[1] *Journ. Brit. Soc. Min. Stud.* Vol. XXII. 1900, p. 105.

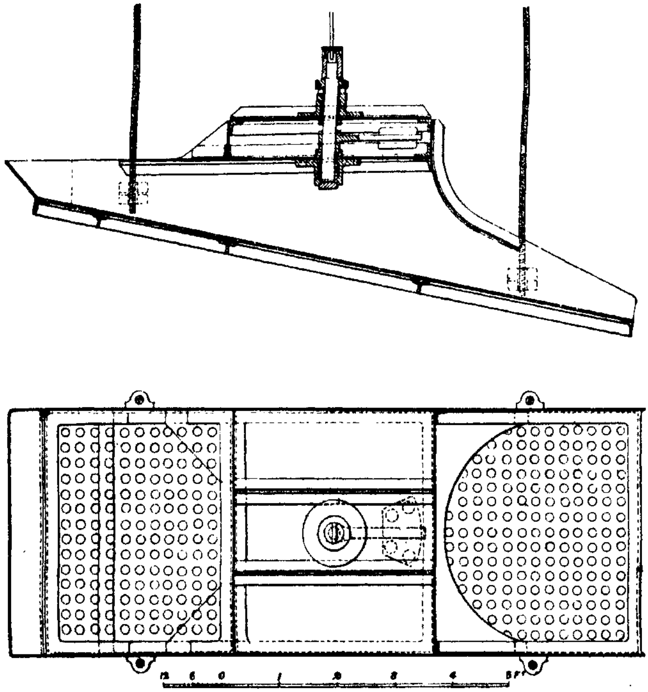

Fig. 35. Large vibromotor screen.

to its inclined surface a particle placed on the screen will descer slowly, describing a series of nearly complete ellipses, and thus trav over a large portion of the surface of the screen, so that thorough efficient screening is obtained. By means of this mechanism, mor over, all the power is utilised in moving the screen, and practically n objectionable vibration is communicated to the framework or buildin in which the screen is contained. A small screen using a simple for of this appliance is shewn in Fig. 34; this is intended for dealin with iron ore at the rate of 2 tons per hour and using 1½ to 1¾ H.P. A similar form[1] may be applied to coal screening; the screen is 6 feet

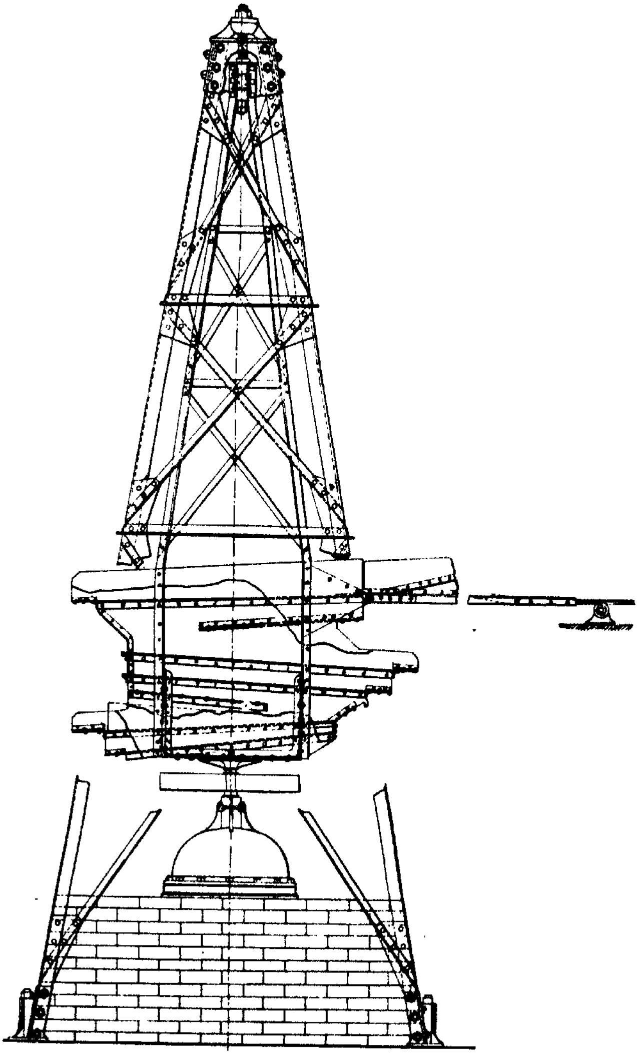

Fig. 36. Karlik pendulum screen.

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[1] *Trans. Fed. Inst. M. E.*, "Coal-washing plant at the Wirral Colliery," by James Platt, Vol. XL 1895-96, p. 55.

by 4 feet and weighs 3½ cwt.; the shaft makes 250 to 350 revolutions per minute and the screen is capable of screening 25 tons of coal per hour through a ⅞ inch mesh, using about 2 H.P. Fig. 35 represents a heavier screen to treat 30 tons of coal per hour, using 3½ to 4 H.P., and with the driving shaft making 350 revolutions per minute. In this screen the vibromotor is in the middle instead of at the upper end, and consists of an arm along which a weight slides instead of an eccentric disc. The mode of action is obviously identical.

Karlik's Pendulum screen. This screen, made by Messrs Schüchtermann and Kremer of Dortmund, is in use at several Westphalian collieries. Its construction is shewn in Fig. 36; it consists of a nest of complex screens, usually containing 3 or 4 screens, suspended by iron rods from a pyramidal framework of iron; the suspension is a simple ball and socket joint, giving great freedom of motion. One side of the nest of screens carries a tray into which the coal tubs are tippled; this tray is prolonged into a girder running on rollers which steadies the movement of the screen. The driving mechanism is underneath the nest of screens, and consists of a short vertical shaft driven by a belt and properly supported on a step bearing. To the upper end of the shaft is keyed a horizontal disc forming a crank, the pin of which actuates the screen. This mechanism is shewn in Fig. 37. The Karlik screen is capable of treating from 25 to 50 tons of coal per hour, and requires 4 to 8 H.P. to drive it.

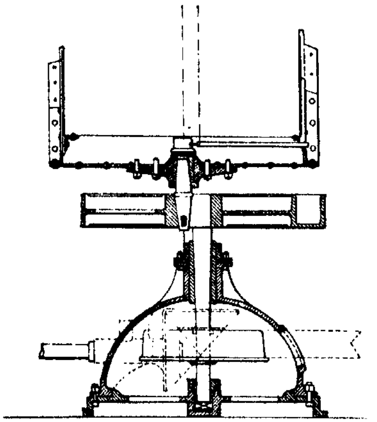
Fig. 37. Driving mechanism of Karlik screen.

Coxe screen. The single Coxe screen[1] consists, like the last, of a nest of screens one above the other, moved also by a crank placed below the screen, keyed on the end of a short vertical shaft. Instead of being suspended, it is carried upon 4 double cones which give it perfect freedom of motion in a horizontal plane, the only condition being that the angle of the generatrix of the cone ABC, Fig. 38, shall be such that $\cos ABC = \frac{\text{throw of crank}}{\text{diameter of base of cone}}$; for coal screens the throw of the crank is usually about 2 inches. In order to prevent these double cones from flying out or shifting from their places they must be guided so as to keep in a definite track; various methods of doing this are shewn in the Figures, the mechanism being shewn in Fig. 39. The complete screen is shewn in perspective in Fig. 40, and it will be noted that the eccentric disc is counterbalanced against the centrifugal action of the screen box, the centre of gravity of which is necessarily not in the centre of rotation. The result of the above described method of suspension and gyration is that each portion of the screening surface describes a circle, but that no two of these circles have the same centre. These screens are largely used in the anthracite region of Pennsylvania; a favourite size is 4 feet by 6 feet screen area for each screen, there being from 2 to 6 screens in one box; the best speed for working is about 140 to 145 revolutions per minute.

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[1] *Trans. Amer. Inst. Min. Eng.*, "The Iron Breaker at Drifton," by Eckley B. Coxe, vol. XIX. 1890-91, p. 398.

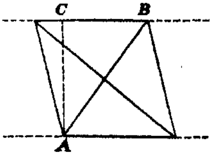

Fig. 38. Diagram of cone of Coxe gyrating screen.

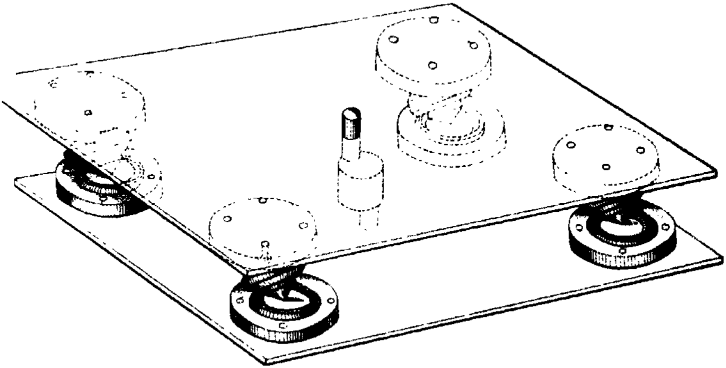
Fig. 39. Mechanism of Coxe gyrating screen.

Coxe screens have been adopted at St Eloy[1] for bituminous coal

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[1] *Bull. Soc. Ind. Min.*, "Houillères de Saint-Eloy," by M. de Morgues, Series III. Vol. XI. 1897, p. 745.

The area of the screen is 5 feet 5 inches by 3 feet 3 inches; the depth of the box is from 1 foot 9½ inches to 2 feet 2 inches, and it contains 3 screens; the top one is punched with holes 2 to 2¾ inches diameter and is set at a slope of 4 per cent.; the next has holes 1⅛ to 1¼ inches diameter and a 5 per cent. slope, and the bottom one holes ½ to ⅔ inch diameter and a 6 per cent. slope. Four sizes of coal are thus made. The screen runs at 155 revolutions per minute and is capable of treating 25 tons per hour.

At Blanzy[1] a screen about 4 feet by 6 feet was found capable of treating 25 tons of coal per hour, making five sizes, the apertures in the screens ranging from 0·5 inch to 0·1 inch, whilst a screen 5 feet by 6 feet, but making five sizes ranging from 4¾ inches to 2 inches, could treat 80 tons in the same time, both screens making 140 revolutions per minute. These screens have also been used for coal in other parts of France and in Germany, and for zinc ores in Algeria.

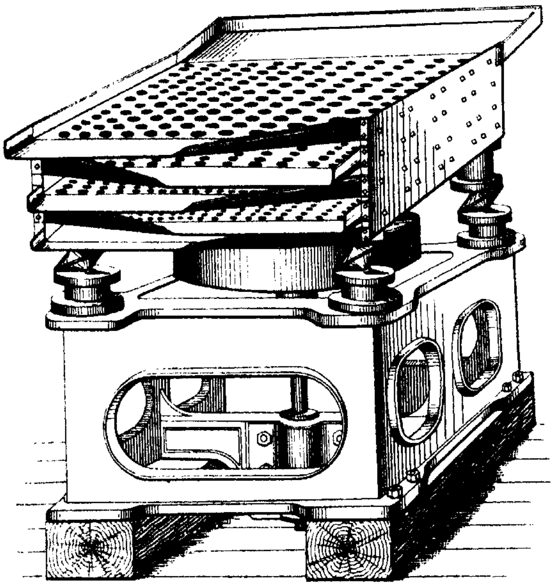

Fig. 40. Perspective view of single Coxe screen.

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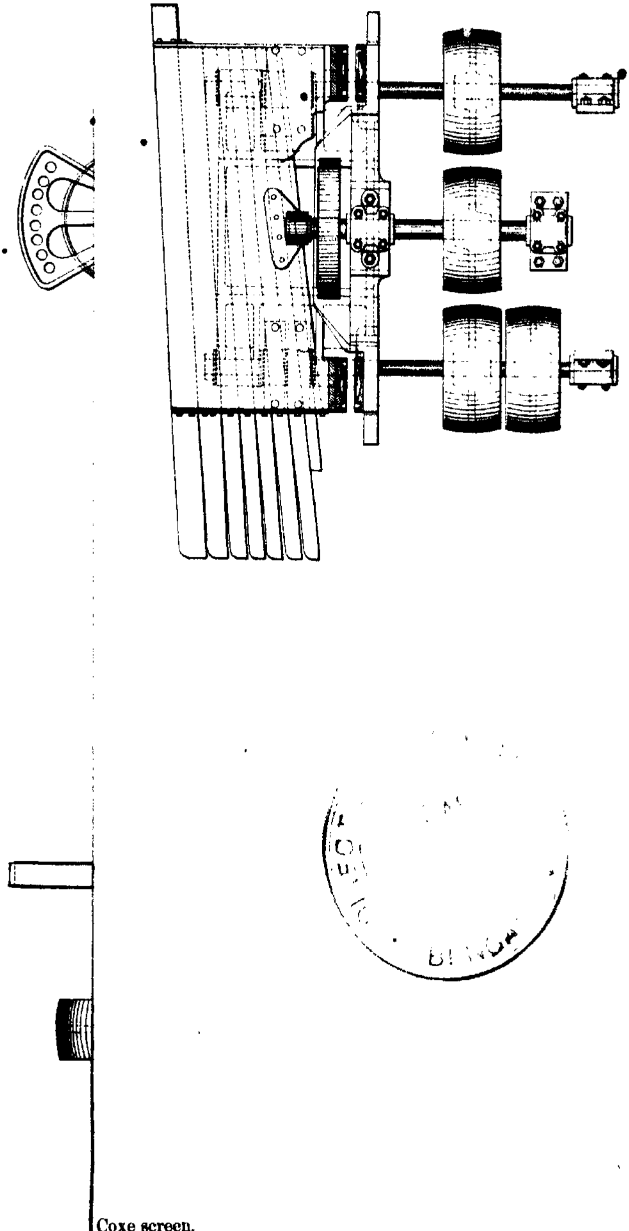
[1] *Rev. Univ. des Mines*, "Préparation mécanique des Minerais," by Aug. Gillon, Ser. III. Vol. XIX. 1892, p. 133.



Coxe screen.

Double Coxe screens have also been used in Pennsylvania; as shewn in Fig. 41, they consist practically of two screens of the first type set side by side, the motive mechanism consisting of two vertical shafts arranged between them, with two eccentrics on each shaft. By this arrangement the lateral pull of the screens is to a great extent balanced and the screens work more smoothly than do the single ones; the whole of the centrifugal force cannot, however, be completely balanced in this way, but this is thoroughly effected by means of two light shafts on which counterpoises are keyed, placed at either side of the structure and driven by means of a belt from the shafts that carry the driving eccentrics. Thus arranged the screens run very smoothly, so much so that nine sizes of coal can be made in one nest; their area is 5 feet by 6 feet and the inclination of the screens range from 1 per cent. for the coarsest size ($5\frac{1}{4}$ inches) to 8 per cent. for the finest ($\frac{1}{16}$ inch).

The Klonne screen is not dissimilar to the last; it consists of a nest of screens moved by a crank, the screens being in this case carried on four segments of spheres upon which rest brackets bolted to the box in which the screens lie; it is said that with screens of an area of 7 square feet the appliance is capable of treating 100 tons of coals per hour.

(γ) **Rotating screens.** These screens are generally spoken of as **Trommels**, a German word (meaning drum) that has passed into pretty general acceptation; they consist of either cylindrical or conical screening surfaces (exceptionally prismatic or pyramidal) rotating about horizontal or slightly inclined axes. The trommel differs from all the other types of screen that have hitherto been considered inasmuch as with the latter the entire screening surface is in continual use, whilst with the trommel only a small proportion is in action at one time; for instance, only about 20 inches of a trommel 5 feet in diameter will be covered under ordinary circumstances, or say under 10 per cent. of the entire surface.

The angle of inclination of the screening surface of the trommel must always be considerably less than the angle of repose of the mineral, so as to prevent the latter rolling straight down it; the angle must however be sufficient to produce a forward motion of the mineral.

In the conical trommel the shaft is horizontal, in the cylindrical trommel it is inclined; in the former case the mineral to be screened enters at the smaller end, in the latter at the higher, and thus in both cases is moved gradually down the inclined plane formed by the trommel surface. In the cylindrical trommel a particle of mineral placed, say in the central line of the trommel at the top end, will be carried up the side of the trommel, by the revolution of the latter, until it reaches such a height on the side that the slope of the screen to the horizontal exceeds the angle of repose sufficiently to cause the particle to glide downwards, which it will do, of course, down the steepest path until the momentum with which it commenced its fall is exhausted, when it will again be carried up, and so on; it thus pursues a zigzag course along the side of the trommel until it is finally discharged. The motion of a particle along a conical trommel is similar, the chief difference being that in the conical trommel the

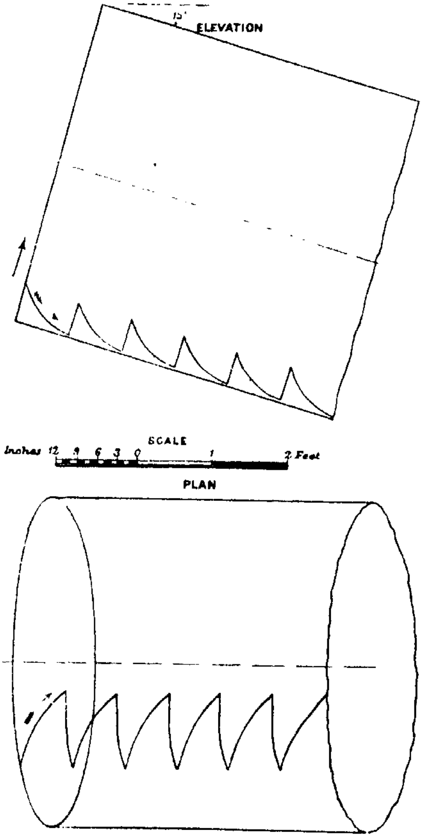


Fig. 42. Path of a particle inside a cylindrical trommel.

particle will be carried up in a vertical plane, whereas in the cylindrical trommel it will ascend in a plane perpendicular to the axis and therefore inclined to the vertical; hence the rate of progression of a particle in a cylindrical trommel will be more rapid than in a conical one at any point at which the two trommels have the same diameter, provided that the inclination of the axis of the one is equal to the inclination of the generatrix of the other. The theoretical paths that would be taken by individual particles along the inner surfaces of trommels are shewn diagrammatically in Figs. 42 and 43, the former representing the cylindrical and the latter the conical trommel. In practice, of course, a number of particles are introduced simultaneously, and the mass will move forwards with a slight zigzag motion, occupying a portion of the circumference of the trommel, which can be approximately determined by the following method, for which I am indebted to Prof. R. M. Ferrier, M.Sc. :—

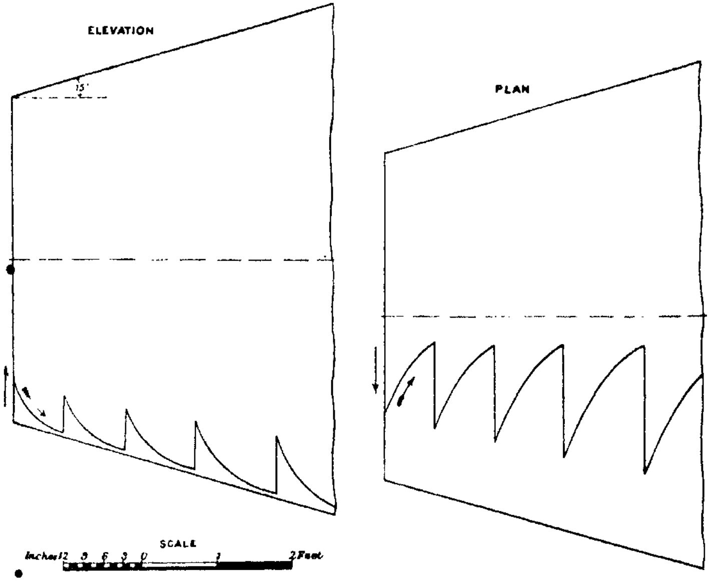


Fig. 43. Path of a particle inside a conical trommel.

In Fig. 44 let *KM* be a section of part of the surface of the trommel, *O* being the centre, and the direction of revolution being indicated by the arrow. Any particle *P* of weight *W* placed inside the trommel at *K* will be carried up with the trommel, until slipping commences at a point *L*; the particle will, however, at first be carried upwards by the trommel faster than it slips downwards, and will hence continue to rise till it reaches a point *M*, at which it just ceases to rise and commences to slide down, which it will continue to do until it reaches a point *N*, at which point all its downward motion will have been overcome by friction against the upward moving surface of the trommel and it is again carried upwards. Neglecting the motion that the particle will receive in the direction of the axis of the trommel owing to the inclination of the trommel surface, the motion of the particle

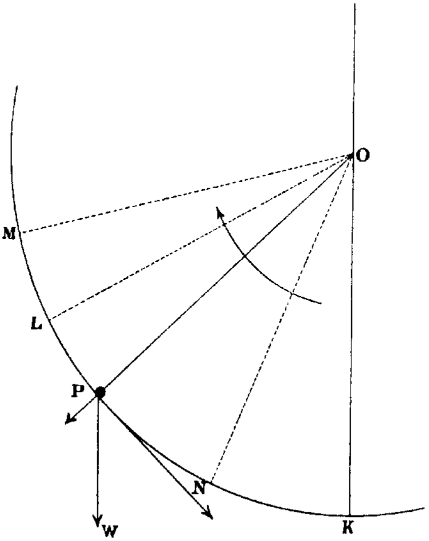


Fig. 44. Diagram of forces acting on particle inside trommel.

in relation to the transverse section of the trommel will consist of oscillations between the points *M* and *N*.

Let *V* be the velocity of the trommel surface in feet per second.

Let *R* be the radius of the trommel in feet.

Let $KOM = \alpha$, $KON = \beta$, $KOL = \theta$.

Let *P* be the position of the particle at any portion of its travel.

Let $KOP = \phi$, and $KP = S$.

Let *v* be the velocity of the particle at any moment relative to the trommel surface.

Let μ_1 be the coefficient of static friction, and μ of sliding friction.

Then the weight W of the particle may be resolved into $W\sin\phi$ acting along the tangent and tending to cause sliding, and $W\cos\phi$ acting along the radius and causing pressure on the inner surface of the trommel. Neglecting centrifugal force, which is a perfectly negligible quantity at the slow speed with which the trommel rotates, the force tending to cause sliding is

$$W(\sin\phi - \mu\cos\phi).$$

The acceleration of the particle $= v\dfrac{dv}{ds} = v\dfrac{dv}{Rd\phi}$; integrating

$$\frac{v^2}{2} = C - gR(\cos\phi + \mu\sin\phi) \quad\ldots\ldots\ldots\ldots\ldots\ldots\ldots(1).$$

At the points M and N the particle is at rest in space, and therefore $v = -V$, or

$$\frac{V^2}{2} = C - gR(\cos\alpha + \mu\sin\alpha) \text{ for } M,$$

and

$$\frac{V^2}{2} = C - gR(\cos\beta + \mu\sin\beta) \text{ for } N,$$

and

$$C = gR(\cos\alpha + \mu\sin\alpha) + \frac{V^2}{2}.$$

Substituting in (1)

$$\frac{v^2}{2} = gR(\cos\alpha + \mu\sin\alpha) + \frac{V^2}{2} - gR(\cos\phi + \mu\sin\phi),$$

$$\frac{v^2}{2gR} = \cos\alpha + \mu\sin\alpha - (\cos\phi + \mu\sin\phi) + \frac{V^2}{2gR} \quad\ldots\ldots\ldots(2).$$

At L, where sliding just begins, $v = 0$, and the inclination of the trommel surface is evidently $\tan^{-1}\mu_1$, this being the point where static friction is just overcome.

Therefore $\theta = \tan^{-1}\mu_1$.

Substituting in (2)

$$0 = \cos\alpha + \mu\sin\alpha - (\cos\theta + \mu\sin\theta) + \frac{V^2}{2gR},$$

$$\cos\alpha + \mu\sin\alpha = \cos\theta + \mu\sin\theta - \frac{V^2}{2gR} \quad\ldots\ldots\ldots\ldots\ldots(3).$$

In this equation the values of θ, μ, V and R are known, and hence it can be completely solved. It obviously leads to two values of α, the higher one of which corresponds to α and the lower to β. It will be noticed that these values are symmetrical to θ, or in other words the particle will rise as far above the angle corresponding to its sliding friction as it will fall below it.

Taking for example the following mean values: trommel 4 feet diameter, making 10 revolutions per minute, when

$$R = 2, \quad V = 2{\cdot}1,$$

$$\mu = \tan^{-1} 25^\circ = 0{\cdot}466, \quad \mu' = \tan^{-1} 31^\circ = 0{\cdot}600,$$

the values for α and β respectively will be about 43° and 7°; in other words the particle will oscillate through an arc of 36°, being 18° on either side of the angle to the vertical equal to $\tan^{-1} \mu$.

In practice of course this result will not be exactly obtained, because not one, but a number of pieces are introduced simultaneously and the trommel is always overcrowded; it may however be considered safe to say the material will never rise above the upper limit indicated by the above equation, though it will often extend down to, or even past the vertical line through the axis of the trommel.

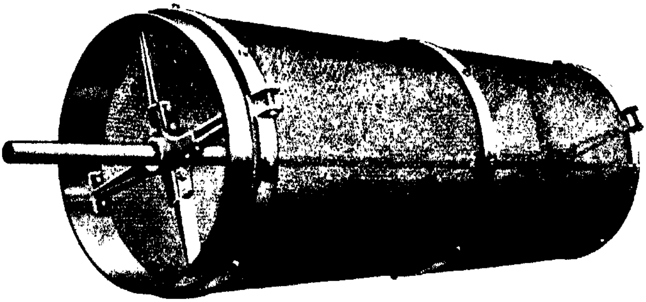


Fig. 45. Cylindrical trommel.

Trommels are constructed in a number of ways, which differ in points of detail, but the essential construction is practically always the same, namely a cylindrical or conical screening surface that is caused to rotate. The screening surface, consisting of wire gauze or perforated sheets, more rarely of bars, is usually kept in its place by straps or bolts, and it may be carried by an axial shaft with spider arms, or else supported on friction rollers. These details of construction may be illustrated by figures of the different types: Fig. 45 is a cylindrical trommel carried on an axial shaft, the screening surface being kept in place by strapping rings; it is made by the Jeffrey Manufacturing Co. The section of such a trommel is shewn in Fig. 46, together with its casing and spouts *A* and *B*, the former delivering the oversize and the latter the undersize. Fig. 47 shews a heavy cylindrical trommel, made by the Allis-Chalmers Co., carried on friction rollers, the screening surface being stiffened by longitudinal straps of angle iron. A simple conical trommel, carried on a horizontal shaft, made by Messrs Bowes, Scott and Western, Ltd., is shewn in Fig. 48, a section of a similar trommel driven however by gearing instead of belting being shewn in Fig. 49. This latter trommel is shewn in the customary sheet iron casing or housing, with spouts; spout *A* is connected with the end of the trommel, and delivers the oversize, whilst spout *B* is connected with the casing, and delivers the undersize. The details of construction of a large

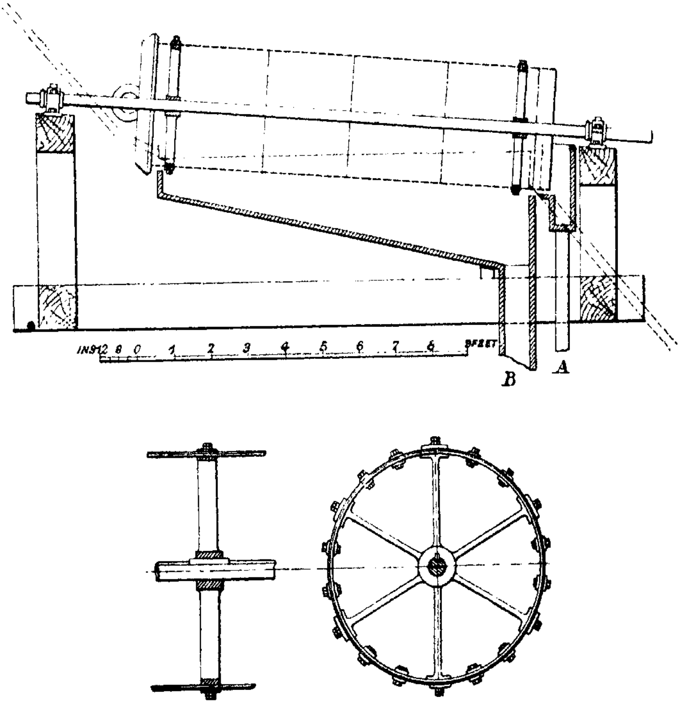


Fig. 46. Sections of cylindrical trommel.

conical trommel are shewn in longitudinal and transverse sections in Figs. 50 and 50^{A}, the form of casing there shewn being the most suitable for very large trommels. Conical trommels are usually mounted on central shafts; exceptionally they are carried in rings which rest on friction rollers. This plan is usually restricted to very light trommels, as in Fig. 51, representing a pattern made by the Jeffrey Manufacturing Co.

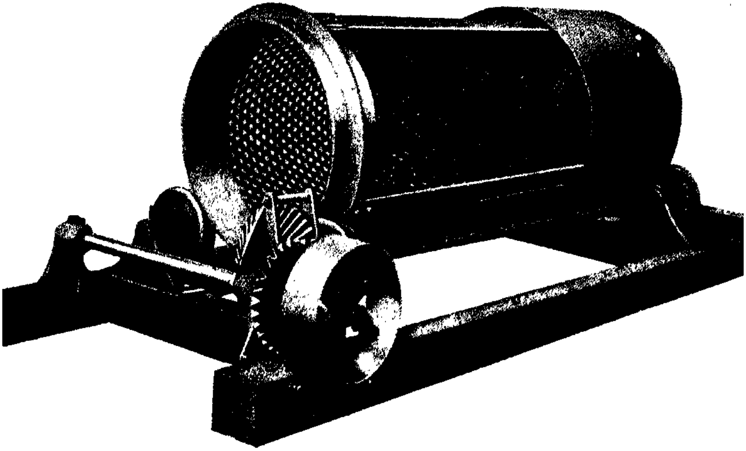


Fig. 47. Perspective view of cylindrical trommel carried on rollers.

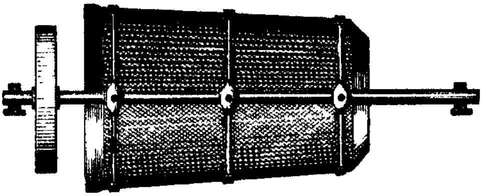


Fig. 48. Belt-driven conical trommel.

The screening surface is sometimes all in one piece, more often in two, three or four sections. When any portion of the screen is damaged the whole section involved has to be removed and replaced; in order to avoid the loss thus incurred, and with the further object of substituting the cheaper form of flat screens for the more expensive curved screen, trommels have been made hexagonal or octagonal in cross-section

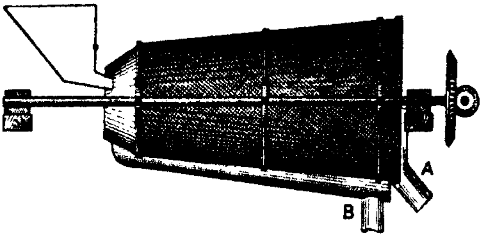


Fig. 49. Gear-driven conical trommel.

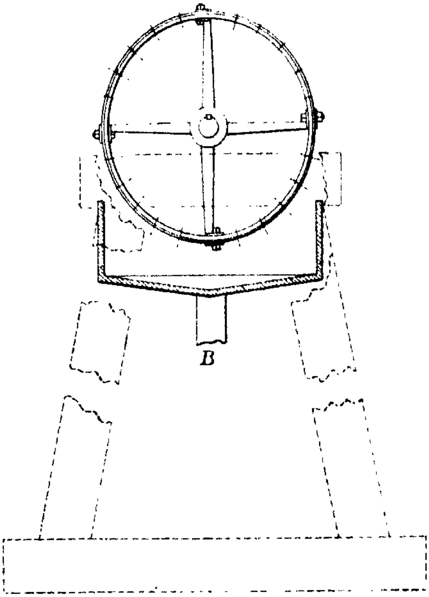


Fig. 50. Transverse section of large conical trommel.

instead of circular; a neat pattern of a hexagonal prismatic trommel is shewn in Fig. 52, which represents a form made by the Jeffrey Manufacturing Co., whilst a similar pyramidal trommel is shewn in

Fig. 53. Such polygonal screens are not however much in favour; in the first place, except in case of an accident, it rarely happens that one portion of the screening surface is destroyed before the whole of it is a good deal worn, so that the loss incurred in replacing the entire screen is more apparent than real. Moreover the angles of the polygon afford lodgement for small stuff, and the larger pieces dropping sharply across

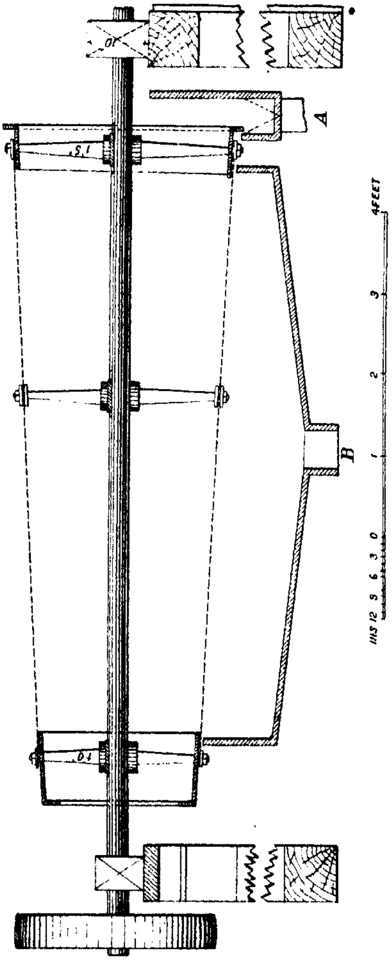


Fig. 50^{a}. Longitudinal section of large conical trommel.

the angle are apt to damage such screens more than does the smoother motion produced by trommels of circular section.

Trommels may be used either for dry or for wet screening; for the latter they are greatly in favour, the material to be screened being run in with water in the form of pulp. It is more often employed for the finer than for the coarser sizes.

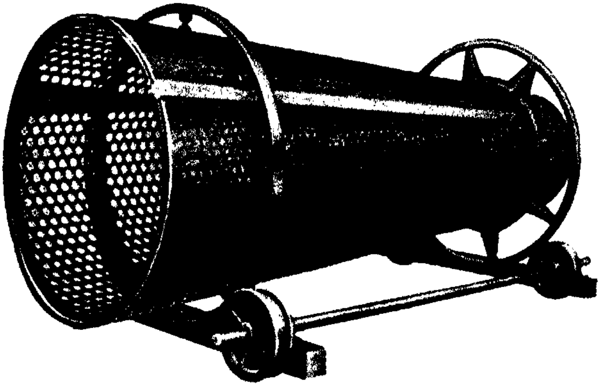


Fig. 51. Conical trommel on rollers; perspective.

The screening surface of the larger trommels is generally made of punched sheets, wire gauze being often used for the finer sizes. Trommels for rough screening have been made of round or flat bars of iron running lengthways and riveted to stout iron rings, but this arrangement is

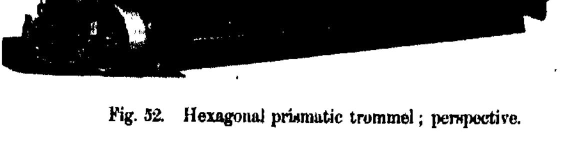


Fig. 52. Hexagonal prismatic trommel; perspective.

not to be recommended; flat screens of one or other of the types already considered are far preferable for all sizes exceeding say 2 inches. On the other hand for fine sizing, and especially for close sizing where a large number of different sizes are required, the trommel does excellent work and is largely used. Sizing by trommels is hence a frequent preliminary to wet dressing operations, and the size of the apertures in successive trommels is regulated by definite arithmetical considerations to be subsequently investigated. For this purpose trommels are grouped in various ways, the mineral to be sized traversing them all in succession; such a series may consist either of groups of simple trommels, of long compound trommels or of concentric complex trommels, the first being the most generally satisfactory. Trommels may be grouped in two ways; either the first may have the coarsest mesh of screen, when the oversize from No. 1 trommel will make the coarsest size, whilst the undersize or all that passes through the screen of No. 1 goes to No. 2, the oversize from which makes the second coarsest size, and so on. In the other system No. 1 has the finest mesh; in this case the undersize from No. 1 screen is the finest size, the oversize from No. 1 goes to No. 2 screen, the undersize from this forming the next finest size, whilst the oversize goes to No. 3 and so on. As a general rule the former is the better method, as in the latter the fine screens are exposed to unnecessary injury from the coarse pieces of ore that pass over it; it is only to be recommended when by far the greater quantity of the material to be screened is fine enough to pass through the finest screens. In grouping trommels, the conical shape presents the advantage that it admits of great compactness, as the trommels can be placed with wide and narrow ends alternately one above the other as indicated in Fig. 54. Cylindrical trommels are difficult to group one above the other on account of the inclination of their shafts. They are best arranged to

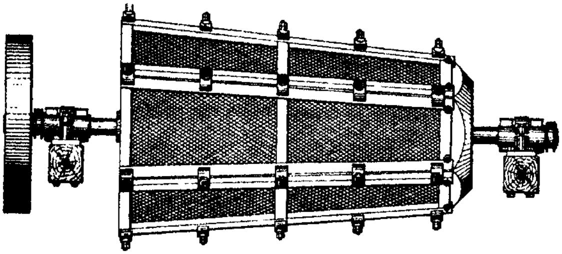


Fig. 53. Pyramidal trommel.

follow each other horizontally as in Fig. 55, each shaft being driven by gearing off its neighbour, or all from one main shaft, which is perhaps the better plan; although this arrangement takes up more space than the former it is often a very convenient one, a row of trommels arranged thus being placed along the side of a dressing mill. This arrangement is equally suitable to cylindrical and to conical trommels, and is the one generally employed; a typical example, where gearing is used, is shewn in elevation in Fig. 55 and in perspective in Fig. 56. Here the material broken to ½ inch passes by the spout *A* to the first trommel

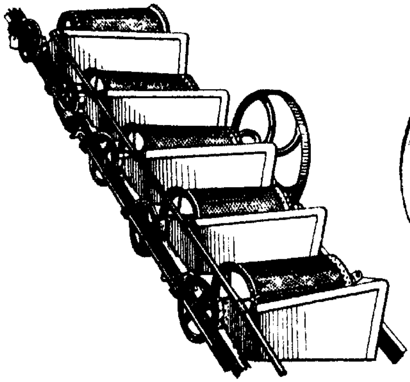


Fig. 54. Set of conical trommels.

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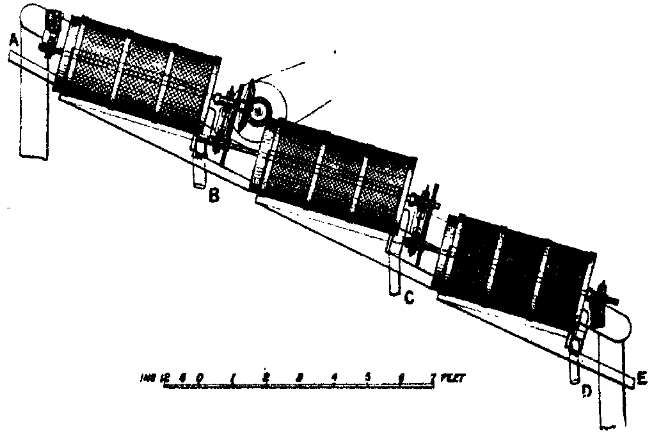

Fig. 55. Set of cylindrical trommels.

with ⅜ inch mesh. The oversize, which is now between ⅜ and ½ inch mesh is delivered by the spout *B*, the undersize passes to No. 2 trommel with ¼ inch mesh; the oversize of this, between ⅜ and ¼ inch, is delivered by the spout *C*, and the undersize goes to the 3rd trommel with ⅛ inch mesh; the oversize from this, between ¼ and ⅛ inch, is discharged at *D*, and all under ⅛ inch at *E*. A somewhat similar arrangement applied to conical trommels is shewn in plan and elevation in Fig. 57 in which the lettering corresponds to that in Fig. 55; in this case each trommel is driven independently by its own pulley from shafting perpendicular to the axes of the trommels; as a general rule such shafts run parallel to the axes.

Compound trommels consist of practically one long trommel successive portions of which are covered with screens of different sizes so that the material is sized in passing along. A triple compound trommel making four sizes is shewn in Fig. 58 and another of much heavier make, without central shaft, and carried on friction rollers in Fig. 59. In these trommels the finest screens must necessarily come first or nearest to the end at which the mineral enters, the oversize from each mesh passing of course to the next coarser. To avoid the serious wear and tear that may thus be caused to the finer screens, the first part of the trommel may be somewhat larger in diameter and may be fitted with a coarse inner screen to keep the rougher material from injuring the finest screens, the oversize from both inner and outer screens passing on to the second part of the trommel. Such a trommel is shewn

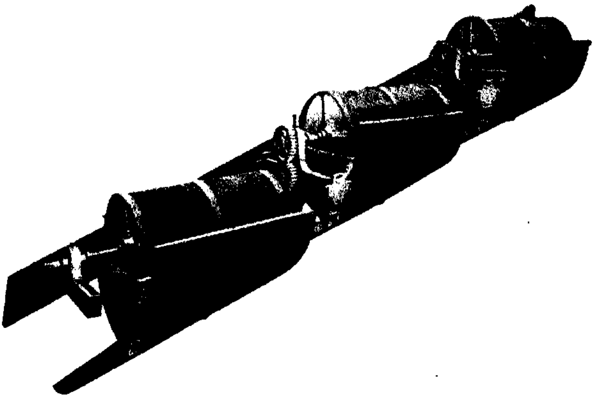

Fig. 56. Set of cylindrical trommels.

in section in Fig. 60[1], where *AA* is the inner protecting screen; it is used for screening coals in the Ruhr district, and has a capacity of 40 tons per hour. It will be noticed that the very long compound trommel shewn consists in reality of two trommels placed end to end,

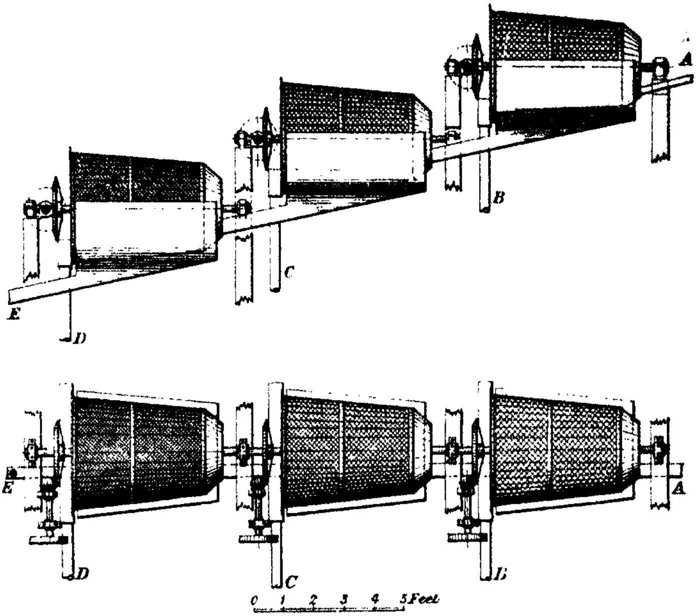

Fig. 57. Set of conical trommels. Plan and elevation.

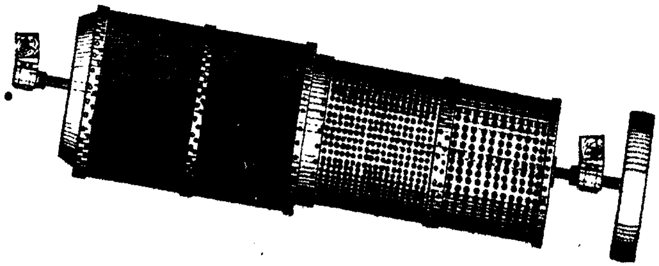

Fig. 58. Compound trommel.

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[1] *Trans. N. E. I. M. E.* Vol. xxviii. 1878–79, p. 163.

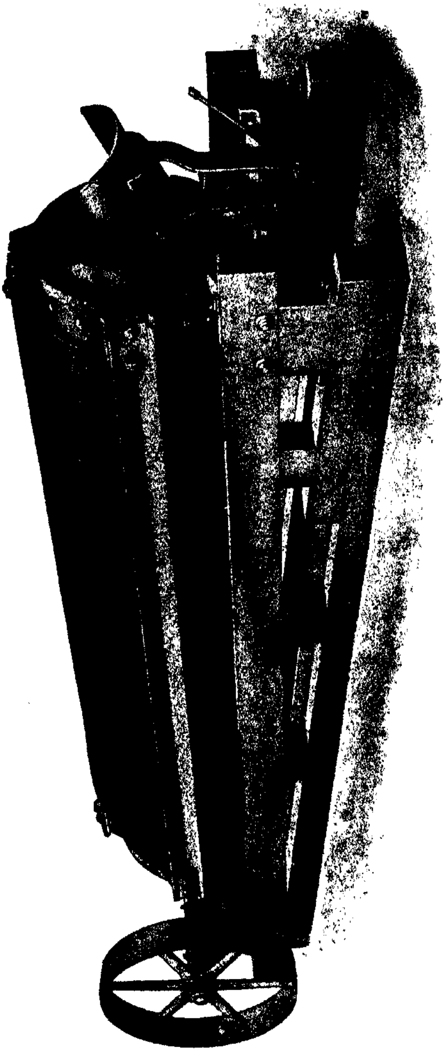

Fig. 59. Compound trommel. Perspective.

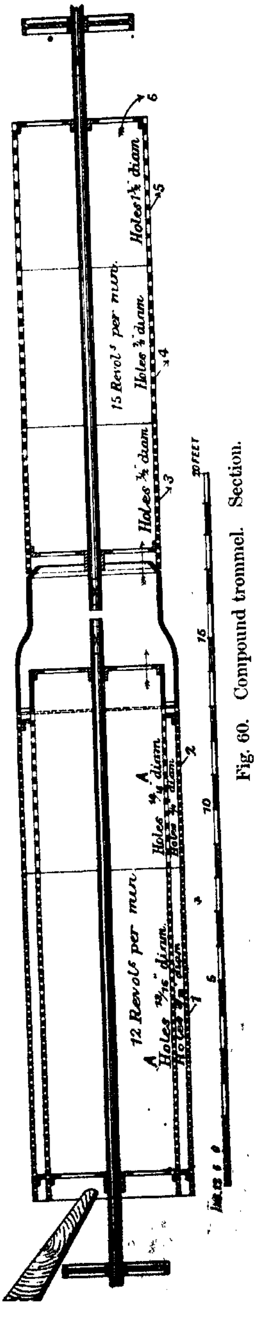

Fig. 60. Compound trommel. Section.

but driven at slightly different speeds, so as to keep the peripheral velocity of both parts equal. Compound trommels are not however very satisfactory when accurate sizing is required, and have indeed little except their cheapness to recommend them.

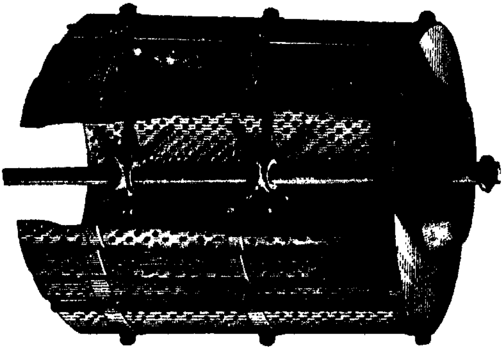

Fig. 61. Complex cylindrical trommel.

In *complex trommels* the innermost screen is necessarily the coarsest. They are compact and convenient in use, the great objection to them being that when one of the inner screens needs replacing the whole apparatus has to be taken to pieces. A complex cylindrical trommel is shewn in Fig. 61, and a double complex conical trommel is shewn in Fig. 62. Sometimes complex conical trommels are arranged with the slopes of the screening surfaces in opposite directions instead of parallel. A large complex cylindrical trommel designed for screening coal by the Humboldt Engineering Company is shewn in Fig. 63; the spaces between each pair of screens are furnished with spiral divisions terminating in

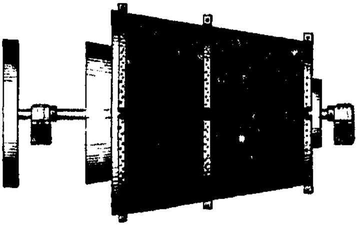

Fig. 62. Complex conical trommel.

spouts so as to deliver the screened mineral, although the axis is horizontal. The innermost screen is prolonged in the form of a sheet iron drum, also furnished with a spiral trough, so that only a determinate

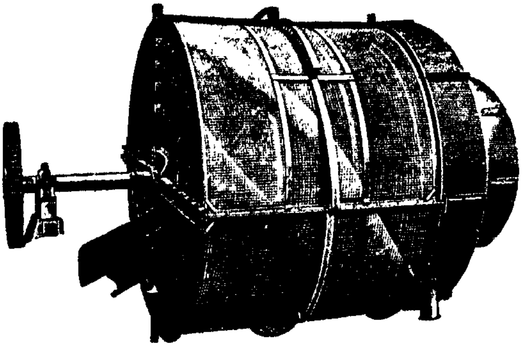

Fig. 63. Humboldt complex trommel. Perspective.

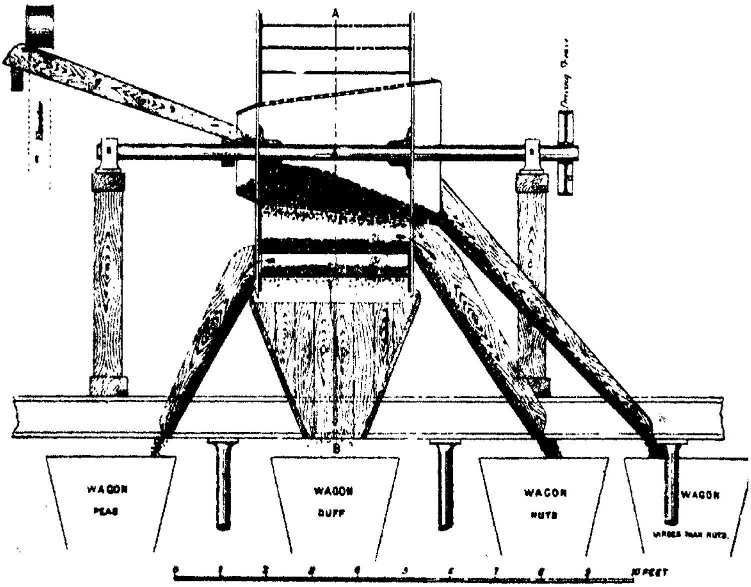

Fig. 64. Schmitt's spiral trommel. Longitudinal section.

quantity of coal is fed into the trommel proper at each revolution; by this means overcrowding of the screens, entailing imperfect screening and breakage of the coals, is avoided. The trommel shewn has five screens 8 feet 3 inches long, the diameter of the outside screen being 10 feet 2 inches, and length over all about 18 feet; the trommel complete weighs about 8 tons and has a capacity of 100 tons of coal per hour.

A somewhat similar machine is Schmitt's[1] Spiral Trommel; this consists of a perforated strip of metal wound round in a spiral as shewn in Figs. 64 and 64ᵃ. The inner portion consists of an ordinary conical trommel round which the spiral is arranged; parts of the spiral are left blank and at these points divisions are put in that separate the different screens from each other. The spiral strip thus forms a complex cylindrical trommel, owing to the special construction of which the coal travels forward although the axis is horizontal. All the mineral that passes through the inner conical screen drops into the next division of the spiral, in which it is carried round and screened, all that passes through falling into the second turn of the spiral and so on;

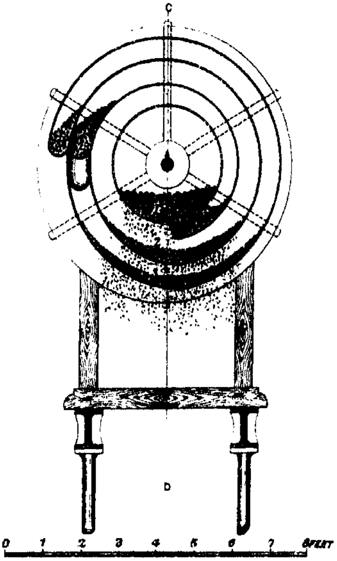

Fig. 64ᵃ. Schmitt's spiral trommel. Transverse section.

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[1] *N. E. I. Min. Eng.*, "Schmitt's Spiral Revolving Screen," by D. P. Morison, Vol. XXVIII. 1878–79, p. 183.

the oversize from each division is discharged by a separate spout. The trommel illustrated is made from a strip 54 feet long by 4 feet wide, and is itself 4 feet long and about 8 feet in diameter; it is driven at 10 to 12 revolutions per minute and will treat 40 tons of coal per hour, the perforations being respectively 1½ ins., ¾ in., ½ in., ⅜ in., 3/16 in. in diameter. It is claimed that this trommel is cheaper and more compact than the ordinary type, strong, easily repaired and economical of power because the load in it is distributed instead of being concentrated at the bottom.

Dimensions, capacity and power consumption of trommels vary within very broad limits according to the material to be treated and its fineness. Lengths are usually between 3 and 12 feet and diameters between 2 and 8 feet; the driving power may range from ⅛ to 1½ H.P. and the capacity from say 2 tons up to 50 tons per hour. The rate of revolution of a trommel should be such as to give a peripheral velocity of 60 to 180 feet per minute working dry and about one half as much when working wet. An ordinary set of 7 trommels about 3 feet in diameter and 5 feet long would size about 7 tons of average crushed material per hour, using about 3½ H.P.; it would weigh about 5 tons and cost about £200 to £250.

In screening the finer sizes care must be taken that the screening surface does not get choked; to prevent this punched plates should be put on with the smaller diameter of the hole to the inside of the trommel as already explained, page 15. An arrangement, suggested by Krom and sometimes adopted, consists of sliding weights upon two opposite spokes, such weights being allowed a play of a foot or so, between strong collars forged upon the spokes. At each semi-revolution the weights will slide downwards and strike sharply against the collars, thus jarring the trommel and helping to clear the screens. In screening wet material, which is often necessary when the mineral to be treated has been crushed wet, the trommel is either allowed to wade in water to a depth equal to about ¼ or ⅓ of its radius, or else a number of jets of water are arranged to spray upon the mineral in the trommel. This is sometimes done by mounting the trommel upon a hollow perforated shaft, through which water is forced. It is however better to have a number of jets under considerable pressure playing upon the top of the trommel, as this tends to clear the screen better than when the jets are directed from inside outwards. It cannot be too often insisted on that, whilst dry screening and wet screening are both quite feasible, successful screening of fine material that is neither quite dry nor thoroughly wet, but merely moist, is an impossibility. It is indeed often necessary to dry finely crushed mineral before attempting to screen it, the only alternative being wet screening in either running or standing water. In coarse screening, say over ¼ inch, this precaution is no longer so essential, though always advisable when possible. In screening soft coals that make much dust, it is sometimes necessary to spray water upon them to keep the dust down; it must not be forgotten that fine coal dust suspended in air forms an explosive mixture and that accidents have been known to occur in coal dressing plants through the ignition of such suspended coal dust.

It is difficult with trommels to size to less than about $\frac{1}{16}$ inch mesh, as the screen is very apt to choke with sizes finer than this. A form of screen specially intended for very fine screening has lately been brought out by Messrs Fraser and Chalmers, Ltd. It is known as the **Callow Screen**, and consists of an endless belt of woven wire screening, 2 feet wide, travelling over a pair of drums 18 inches in diameter, set at about 4 feet 6 inches centres, over which the screening belt runs continuously at a speed of 25 feet to 125 feet per minute. The screen works wet, and for the finer sizes (below 60 mesh) its action is supplemented by a shaking spray; the capacity of a duplex machine, consisting of two 24 inch belts, is given as ranging from 250 tons per 24 hours on 20 mesh material down to 50 tons per 24 hours on 120 mesh material. The appliance has been so recently introduced that no definite information about its work is available. A similar screen has been devised by Mr Stanley of the Nuneaton works.

CHAPTER III.

SORTING AND WASHING.

Principles. When mineral is brought up from the mine it often happens that it is mixed with a great amount of barren or worthless material which may be removed by simple hand-picking, the appearance or the "feel" of the minerals being the sole means employed in such selection; sometimes in the same way minerals may be classified into several different qualities or kinds. For example, in Cornwall, copper ores are picked out from tin ores and set aside for separate treatment; in many lead mining districts pure galena ("potter ore") is picked out from the lead ores mixed with gangue or from mixed lead and zinc ores, which have to undergo more elaborate processes of dressing; on the Witwatersrand the gold-bearing banket is picked out from the barren quartzite which is broken down with it; in most coal mining districts shale is picked out from the good coal; in the Hamilton district, splint coal, house coal and cannel coal are in places got all together in one seam and are subsequently picked out; in the Cleveland district the worthless "dogger" and shale are picked out from the ironstone; at Rio Tinto various grades are made by sorting, such as quartzose ore, smelting ore, leaching ore, export ore; at Gellivara, magnetite is sorted into five different grades by the eye alone, according to the amount of apatite it is seen to contain, this sorting being afterwards checked by chemical analysis.

All the above are typical examples of sorting or picking, some of them being final inasmuch as the sorted mineral is at once ready for the market, whilst others are merely preparatory to further dressing processes.

Some minerals are got covered with clay, mud, or dirt, to such an extent as to render sorting impossible, or in other cases to decrease greatly their market value; in such cases the covering of dirt has to be washed off either as a preliminary to sorting, or else as a preparation for the market. The former treatment is applied to a large number of complex veinstuffs, such as those carrying copper, lead, zinc, etc.; the latter is most largely applied to iron ores, to certain calamines, phosphate rocks, etc., the objectionable matter generally consisting of clay which is often tough and tenacious. Sorting and washing are at times carried on nearly or quite simultaneously on the same appliances, in other cases special apparatus is used for the one or the other process. The operations are however often so closely connected that they can be conveniently considered in the same chapter; those appliances that are used for washing only will be taken last, those used for sorting or for sorting and washing coming first. Sorting is often combined with hand-breaking, and can, in most cases, only be performed by the aid of a certain amount of breaking; the subject of hand-breaking will be deferred to a subsequent chapter, when the hand tools used for breaking with the object of sorting will also be considered.

Sorting is performed either upon floors or fixed tables, or upon movable surfaces; in the former the workpeople engaged turn over the mineral, separate it into the various classes, and make heaps of these, which are then wheeled or carried away either into railway waggons for shipment, or else to bins, etc., for further treatment. When moving appliances are employed the mineral is carried mechanically past the sorter, who picks out the various kinds. The former method evidently needs far more manual work than the latter, but the mineral is also likely to be more thoroughly picked, as every lump has to be turned over by hand; its use is therefore indicated in places where the price of manual labour is low compared to the wages paid to skilled artisans, where stores, lubricants and machinery repairs are relatively expensive, and where close sorting is an object of special importance. Sorting is generally not heavy work, so that boys, girls and old men are often employed as sorters, the former classes being especially satisfactory. It is usual to arrange matters so that in a big sorting establishment the majority of the workers shall be of the above category, whilst there are merely one or two more highly paid men, who can break up and handle the larger lumps, or do any similar exceptionally heavy work.

At mines where only a small output has to be dealt with, the mineral is sorted on a table usually covered with heavy sheet iron, or sometimes a cast iron plate is employed, which may be perforated with holes ½ to ¾ inch in diameter. A very usual and satisfactory arrangement is for the cars as they come from the mine to be tipped upon a grizzley, the angle of which is so flat that the mineral can just not slide down it; upon the mineral as it lies on the grizzley a stream of water is allowed to play, which washes the fine ore and any dirt through the grizzley bars; the fines are shovelled out from time to time and sent to the dressing works. The lump ore is thus effectually washed, and is in good condition for sorting. At the foot of the grizzley, the picking table proper is placed, usually about 3 feet square, and about 2 feet high above the floor. The sorters rake the ore from the grizzley on to the table as required, sort it, and either pile up the various classes in heaps on the floor behind them or throw them into handbarrows, wheelbarrows or small cars, ready for removal to their destinations. A diagrammatic section of such an arrangement is shewn in Fig. 65 in which *AA* are the grizzley bars, and *B* the picking table, this being the usual form in the lead mines in the North of England. In Cornwall a very general arrangement consists of a long table about 3 feet broad and 3 feet high extending along one or both sides of the picking shed (in which *bucking* [see p. 111] is also performed); along the back of the table runs a raised plank, on which the lads or girls bring in the ore in handbarrows.

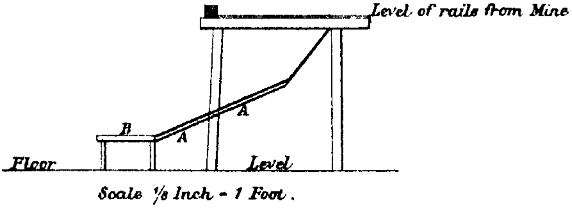

Fig. 65. Diagram of sorting table.

For dealing with larger outputs the picking table is dispensed with and sorting is done on the floor itself. The most convenient arrangement is that shewn in diagrammatic section in Fig. 66 and which is similar to that in use at Rio Tinto, where 200 to 300 men work on a floor some 350 feet in length, sorting 1000 tons per day. The ore comes in by railways on an embankment or trestle gantry, which should be not less than 15 feet above the level of the sorting floor. The different grades are loaded up into other railway waggons, the rails being so disposed that the top of the waggon is flush with or just below the level of the sorting floor.

Fig. 67[1] shews a diagrammatic cross-section of the sorting floor at the Ferreira mine, Witwatersrand. The floor proper is about 11 feet wide, and 70 feet long; the ore after passing over a grizzley drops some 3 feet on to a floor covered with steel plates, where it is washed by means of hose pipes, and the barren quartzite is sorted out from the clean ore; the former is loaded into trucks and run out to the waste dump, the latter is shovelled into a large bin for further treatment.

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[1] "The Witwatersrand Goldfields," by S. J. Truscott, p. 415.

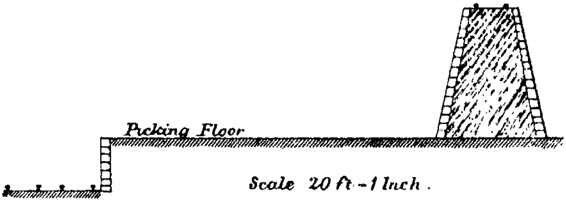
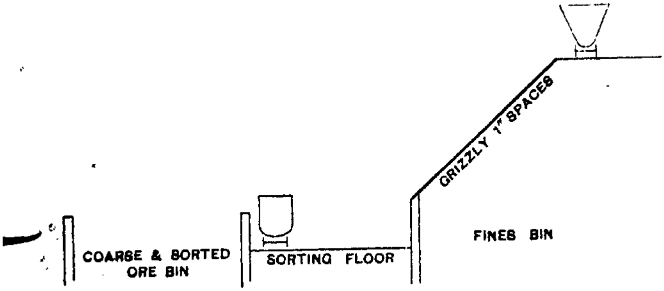


Fig. 66. Sorting floor for large outputs.

Here, as elsewhere, where gold ores are washed, the water used for washing is run into settling pits so as to recover the fine auriferous material which it necessarily carries in suspension. The labour staff on this floor consists of 11 natives and a white foreman, and some 400 tons a day are treated, of which about 36 per cent. is sorted out; the cost is about 6*d.* per ton of crude mineral, of which about 14 per cent. is for supervision, 82 per cent. for native labour, and 4 per cent. for stores.



Scale, 1 inch = 10 feet.

Fig. 67. Diagram of sorting floor at the Ferreira Mine.

Attention must be paid to the paving of these sorting floors. For minerals of small intrinsic value clay or earth beaten hard may answer, but they are better paved with slabs of stone; when the value of the mineral is greater, brick on edge set in mortar or cement may be used, a very hard burnt brick or slag brick being the best for this purpose. Steel or cast iron plates are often used when such minerals as gold or silver ores have to be sorted; concrete or cement is rather soft, and is apt to get cut up by the wheels of wheelbarrows or by breaking minerals on it.

When coal is picked on a fixed support, this usually takes the shape of the grizzley shewn in Fig. 4, p. 18, sorting being performed on the lower part of the screening surface, or on the sheet iron apron in which it terminates. There are usually 2 to 3 pickers to each screen; about 12 tons per hour can be screened and sorted on one of these.

Moving Appliances used for sorting may be either shaking trays, revolving tables or endless belts. All of them depend upon the general principle that the material to be sorted shall be carried in a thin layer past the sorters, who have thus an opportunity of picking out and separating the various classes of mineral from each other. For successful picking it is requisite that the speed be not too great, say about 80 feet per minute on the average, and the layer of mineral so shallow that every piece may be seen; the quantity of mineral that can be sorted in a given time and with a given number of pickers depends entirely upon the proportion of mineral that has to be picked out, and the number of classes that have to be made, and must therefore vary within very wide limits. The width of the stream of mineral that a picker can examine effectively is about 2 feet and should never exceed 2 feet 6 inches.

Shaking trays. These are either like the vibrating screens described on p. 38, or the conveyor screens described on p. 46, except that the screening surface is replaced by a sheet of iron. Sorting to a small extent is also occasionally, though rarely, performed on ordinary vibrating screens. All these vibrating appliances are however unsatisfactory because the distance travelled is short, the layer of mineral is deep, and the jerky to and fro motion is very ill adapted to careful picking. Accordingly shaking trays are but little used; their first cost is low and they take up but little room, but their performance is decidedly unsatisfactory.

Swinging conveyors answer better, but even with these some practice is needed, because the to and fro motion makes the pickers dizzy at first. A Zimmer conveyor in two lengths, the throw of one of which balances that of the other, is shewn in Fig. 68; coal picking has been performed on this conveyor, e.g. at Shipley Collieries, where 40 tons of coal per hour pass over a conveyor having two compartments side by side. Any of the conveyor screens described on page 48 may be used for picking purposes.

Revolving tables. The simplest form of revolving table is shewn in Fig. 69, this being a form manufactured by Messrs Fraser and Chalmers, Ltd. The annular table covered with perforated iron plates is attached by arms to a central vertical shaft which is slowly rotated by means of a worm wheel and tangent screw in the direction shewn by the arrow. The mineral is delivered on to it by means of a shoot, *A*, or a shaking tray, usually after having been screened; it makes about ¾ of a revolution, during which the sorters standing round the table pick off the material to be removed and throw it into bins or trucks standing behind them. Just above the surface of the table a vertical iron plate is fixed diagonally which forms a plough, *B*, sweeping off all the mineral remaining on the table into a bin or shoot, *C*. The table shewn is perforated so that if the mineral lying on it requires washing, the water may drain off through the holes; this is collected in the light iron funnel shewn beneath the table and led off by a trough into settling pits. When used for minerals (such for instance as coal) that do not need washing, the surface of the table is covered with plain plates and the draining funnel beneath it is omitted.

Fig. 70[1] is a diagrammatic plan of another form of table which presents the advantage that pickers can stand both inside and outside it, bins (marked *W* in the plan) being disposed both around the circumference

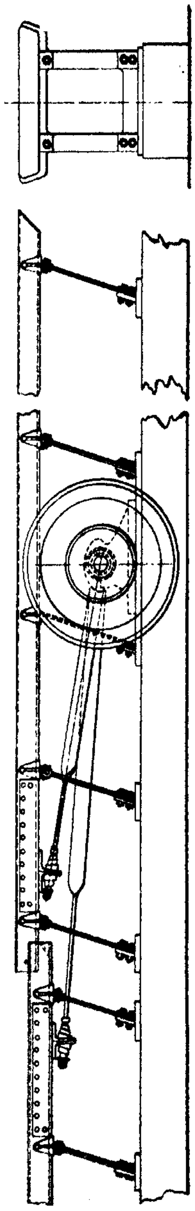


Fig. 68. Zimmer picking conveyor. Side and end elevations.

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[1] "The Witwatersrand Goldfields," by S. J. Truscott, p. 417.

nd inside the annular table. The table is carried on a ring of wheels paced 5 feet apart, which run on a circular rail, and is driven by a pinion

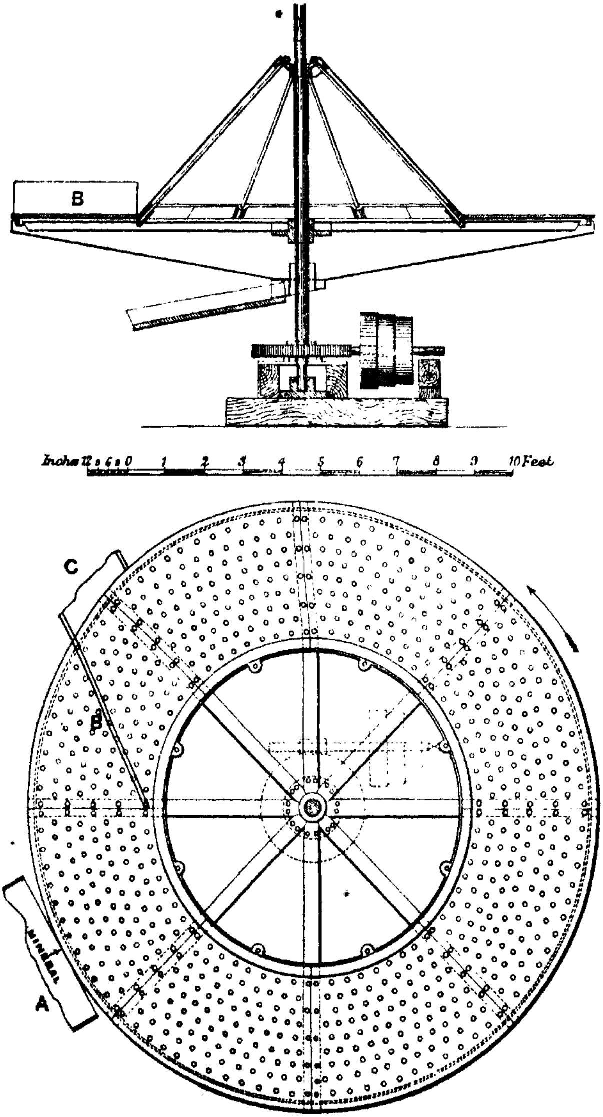
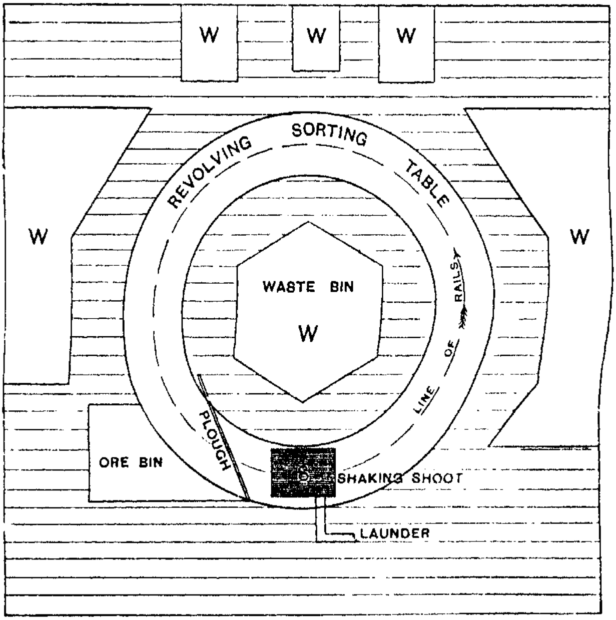


Fig. 69. Revolving picking table on central shaft. Plan and vertical section.

gearing in a circular rack bolted to the underside of the table. It makes a revolution in about 1½ minutes, its peripheral speed being about 55 feet per minute. The table is 4 feet wide and its top consists of a plate of iron ½ inch thick, upon which is an upper removable plate ⅝ inch thick. The table slopes all round to the outer edge, down to a circular launder, which carries the water used in washing the ore into settling tanks. This table is in use at the Crown Reef Mines, Witwatersrand. Usually there are 8 natives at work on this table, only 4 of whom are however, strictly speaking, engaged in sorting; from 650 to 700 tons per day can be sorted on this table, 14 per cent. being picked out. The operating cost appears to be about *4d.* per ton of crude ore for working expenses and about *3d.* per ton for maintenance.



Scale, 1 inch = 10 feet.

Fig. 70. Diagram of picking table at Crown Reef.

Another form is shewn in Fig. 71, differing from the last in that the table travels on rollers which are carried by uprights, so that the rollers do not travel round with the table; this plan appears to be better than the last as the rollers are not so likely to be stopped by pieces of mineral dropping in front of them. Here also *A* is the feed shoot, *B* is the plough, and *C* the delivery shoot for the material remaining on the table. Fig. 72 shews a somewhat similar table, but differing in some details. Such a table is in use at the Cramlington Colliery, the top being made of steel plate ¾ inch thick. It makes a revolution in about 50 seconds, and 7 men and boys are employed as pickers, these standing round the outside of the table only. It takes 3 tubs of coal, say 18 cwt.,

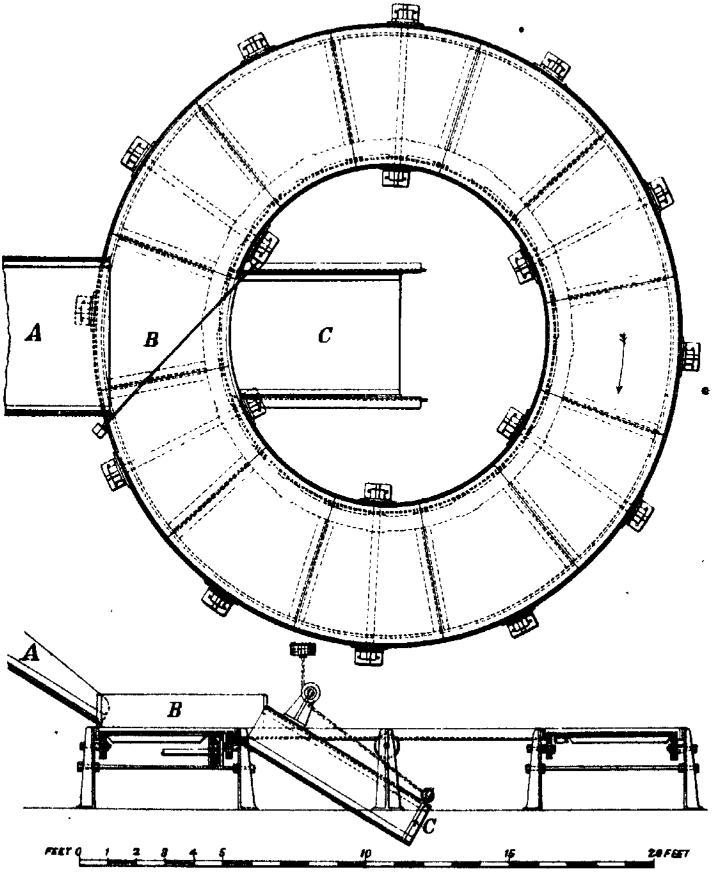


Fig. 71. Picking table. Plan and vertical section.

at a time, thinly spread, from which about 7 per cent. of splint and 3 per cent. of stone are picked out, leaving the remaining 90 per cent. on the table to be swept into the shoot *C* which can be lowered or raised as required to suit the waggons that are being filled. Its capacity is rated at 400 tons in 10 hours and it does very satisfactory work.

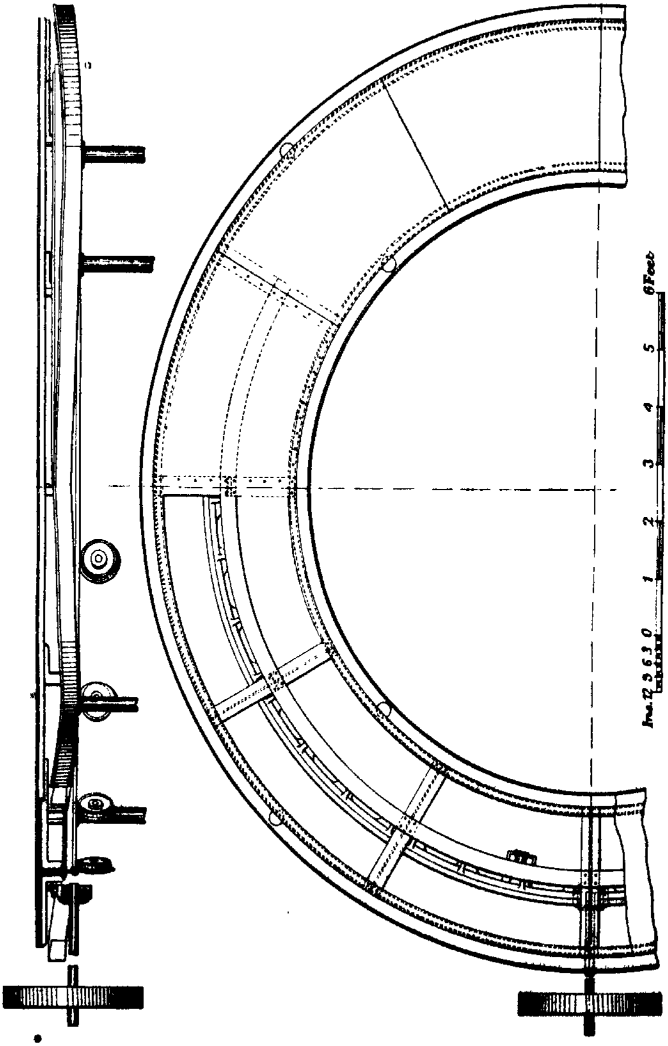


Fig. 72. Picking table. Plan and elevation.

The Humboldt Engineering Company makes similar tables carried on ball bearings instead of on rollers, which work very smoothly. These are made in three sizes, from 10 feet to 16 feet 6 inches in outside diameter, and 2 feet to 3 feet wide, making a revolution in from 2 to 4 minutes and requiring from ½ to 1½ H.P. to drive it.

For larger tables this firm uses the pattern shewn in Fig. 73 in which the table is suspended by an "umbrella" frame from a central vertical post so that pickers can stand both inside and outside the table. This table is made up to 26 feet 3 inches in external diameter, makes a revolution in 4 minutes and takes 2½ H.P. to drive it. These tables are best set up within a circular wall of brickwork or plank which encloses all the space below the annular table; against this wall shoots or pockets are built into which the waste rock, or other material picked off the table, can be thrown. If shoots are used these usually lead into one common spout from which the mineral thus collected is run into a suitable waggon. Sometimes instead of such shoots a shelf is fixed to the circular wall, upon which are placed hand-barrows or boxes to receive the products picked out.

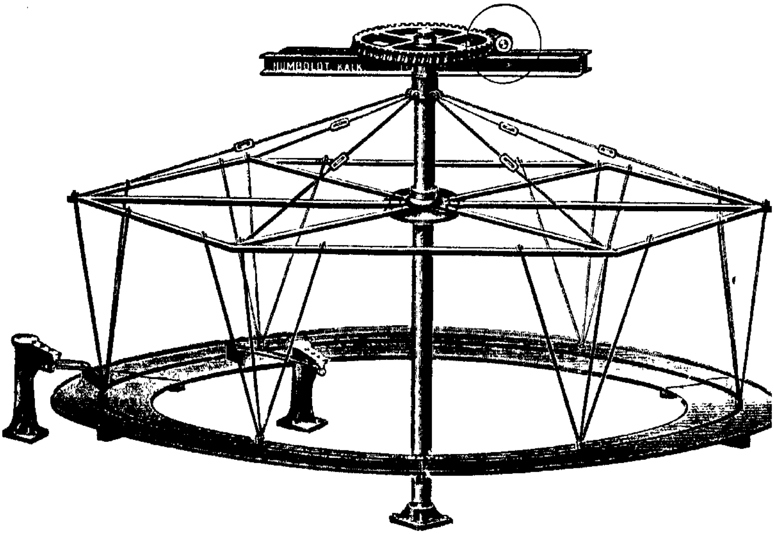


Fig. 73. Large picking table. Perspective.

Another form of table is used at the Consolidated Main Reef, Witwatersrand; this is shewn in Fig. 74[1]. It will be seen to be similar to the first type, Fig. 69, in that it is supported by a central shaft; it is

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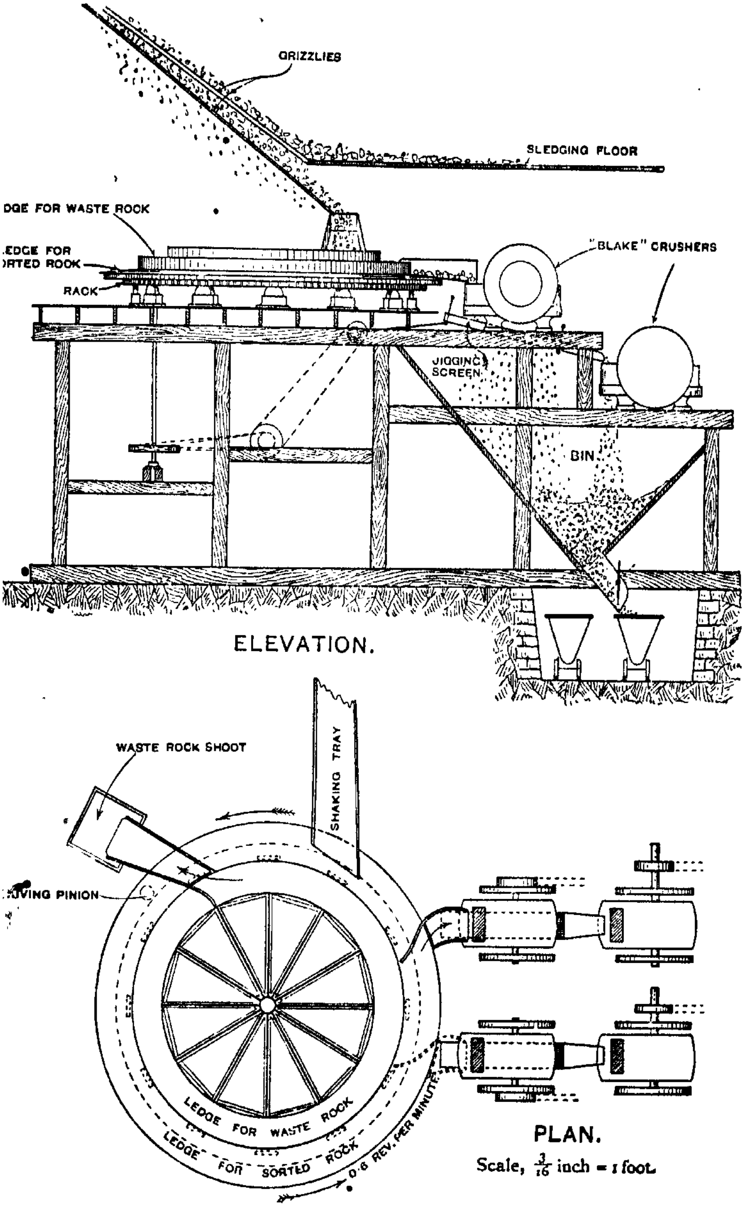
[1] "The Witwatersrand Goldfields," by S. J. Truscott, p. 421.



Scale, $\frac{3}{16}$ inch = 1 foot.

Fig. 74. Double decked picking table.

however driven by a circular rack and pinion, like the table shewn i Fig. 72, making a revolution in 1½ minutes equal to a periphe velocity of 50 feet per minute. Its peculiarity consists in the fac that the table is stepped, consisting thus of a lower main surface an an upper shelf; the ore to be picked is delivered by a shaking tra on to the table proper; the barren rock is picked out and placed o the upper shelf; the clean ore is first swept off by one plough, an the barren rock by a second one, each into its proper receptacle. Th advantage of this double-decked table is that the waste material is ope to the inspection of the foreman just as much as the cleaned mineral, s that he is able to see that no good ore is thrown on to the wast dump. This form of table has been used with much advantage fo coal, both for picking out stone and for sorting coals of different kinds it affords probably greater facilities than any other form of picking appliance for dealing with "laid out" tubs, that is to say with tubs sent up from the pit with an undue proportion of stone and for which deductions are made from the miner's wages. Each tub can be readily kept separate on the main table, and the stone in it placed on the shelf immediately above it.

In special cases, three-decked picking tables have been used when several grades of mineral have to be sorted out. For example in Scandinavia a table of this kind is used for picking an ore consisting of gangue, galena and zinc blende. The broken ore is delivered on to the top deck of the table; the waste rock is picked out and placed on the middle deck, and the lump blende is similarly placed on the lowest deck, the ore that requires further dressing, which consists of a mixture of galena and blende, remaining on the topmost deck.

The quantity of material that can be dealt with on any of these tables varies within very wide limits according to the nature of the work to be done and the amount of picking required.

Picking belts. These are very extensively used, especially for sorting or cleaning coal. As compared with circular tables the latter are more compact, a 24 foot table being for example equal to a 75 foot belt, and going into a much smaller space, where only one is required. A number of tables side by side would however occupy about as much room as an equal number of belts of equivalent length. For moderate plants there is much to be said in favour of revolving tables; direct comparisons are difficult to make, but it appears that the tables should cost considerably less for maintenance and should also require less

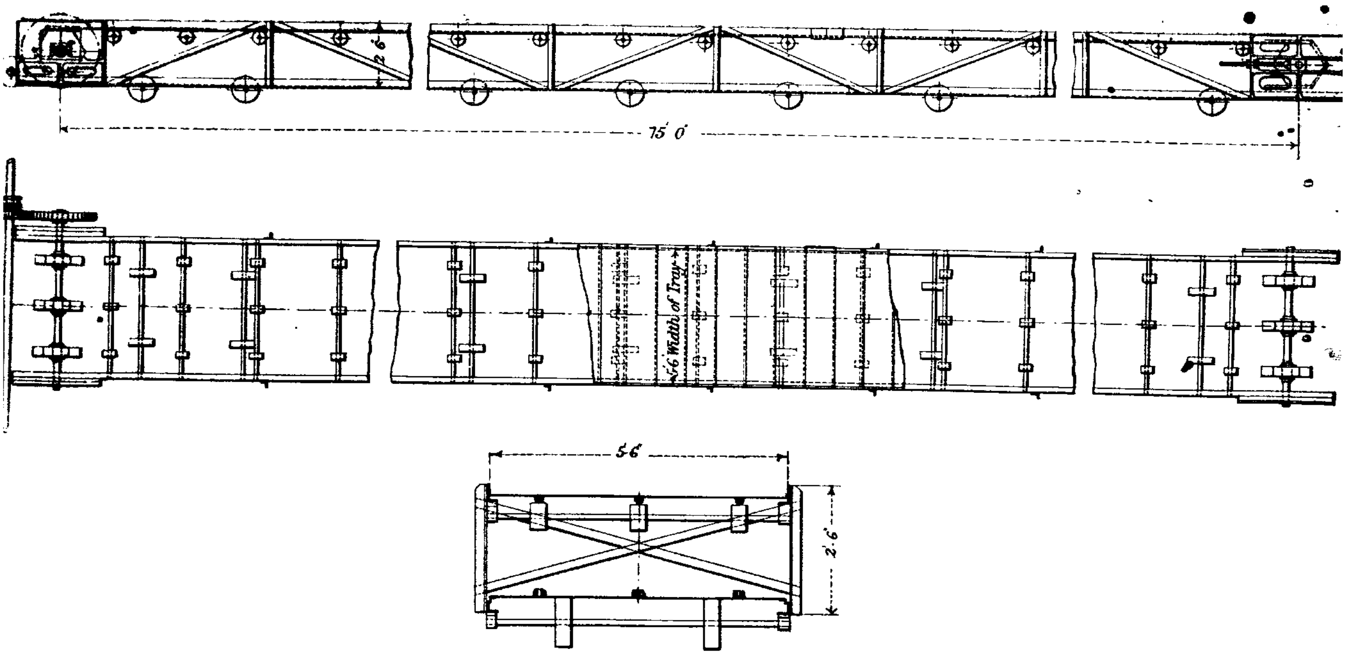


Fig. 75. Picking belt. Plan, side and end elevations.

power, whilst supervision of the pickers is also easier; on the other hand the long narrow belt often suits the general arrangement of colliery heapsteads which are very generally built over part of the railway sidings on which the coal trains are made up, so that the great length is not a serious objection. Belts require less fall than tables and in fact a long belt may even rise a few feet towards the deliver end so as to gain headroom, an advantage that is presented by no othe sorting appliance.

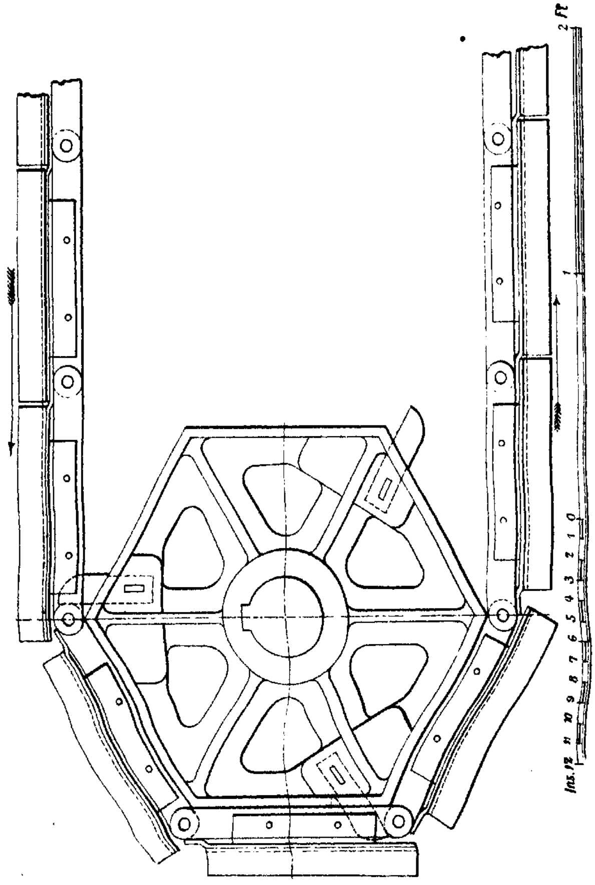


Fig. 76. Driving drum of picking belt.

Picking belts are made of various materials, the most usual for consisting of iron or steel plates 4 to 5 feet long and 1 foot t

1 foot 6 inches wide, which are attached to two, or at times three, endless chains, the length of each link of which is just equal to the width of the plates. These chains pass round a couple of drums supported on either end of an iron or wooden framework that carries also a series of rollers, by means of which the belt is supported. Usually these drums are hexagonal or octagonal, the width of the face of the polygon being equal to that of the plates; such an arrangement gives ample driving power, but some makers prefer a special sproket wheel that gears in the links of the driving chain or else special driving teeth projecting from the drum are used as in Fig. 76. When a polygonal driving drum is used, it should be grooved for the chains to lie in, the plates bearing flat against the faces of the drum. Either end of the belt may be the driving end, but it is preferable whenever possible, more particularly in the case of a very long belt, that it should be driven from the front or delivery end; this will cause the upper side of the belt to be in tension, and thus relieves somewhat the pressure that would otherwise come upon the supporting rollers. The following drum (i.e. the drum that is not being driven) should be carried upon bearings that travel on slides and can be drawn up by means of screws; the tension of the belt can thus be properly adjusted, and any wear that may occur be taken up. The general arrangement of such a picking belt, made by Messrs Cook, Sons and Co. Ltd., is shewn in Fig. 75, which also shews *the framework upon which it is generally supported, the driving drum being shewn on a larger scale in Fig. 76.*

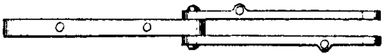


Fig. 77. Chain for picking belt.

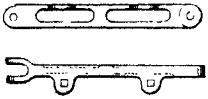


Fig. 78. Cast steel links for chain of picking belt.

The chains are of various types; they occasionally consist of single links connected by pins, but more often of double and single links alternately (Fig. 77[1]); at times cast steel links, one end of which forms a jaw (Fig. 78[1]), are employed; the latter are light and neat, but are more apt to break than the former pattern and can scarcely be repaired. The plates are generally sheet steel, about $\frac{3}{8}$ inch thick, and overlap each other about 1 inch at the edges in the manner shewn in Fig. 79, so that the surface of the belt shall be level; of course the overlap must be so arranged that each plate covers the one that precedes it in the direction of motion, otherwise small pieces of coal would lodge in the space formed when the plates pass over the drum at the delivery end.

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[1] *Trans. Fed. Inst. Min. Eng.*, "Improved Coal Screening and Cleaning," by T. E. Forster and H. Ayton, Vol. I. 1889–90, p. 83.

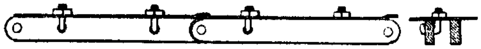

Fig. 79. Hook bolts for plates of picking belt.

The plates are sometimes riveted or bolted direct to the links; the former plan entails the objection that when a plate has to be replaced, cutting the rivets and putting in new ones takes a good deal of time, the latter that the heads of the bolts project above the belt. The same objection applies to the use of the hook bolts sometimes employed, see Fig. 79[1]. A better plan consists in riveting short lengths of angle iron to the plates, and bolting the angle irons to the links as in Fig. 80[1].

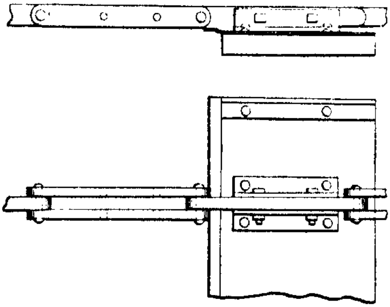

Fig. 80. Angle iron attachment for plates of picking belt.

The belts are supported by rollers at intervals of about 3 feet; those carrying the empty return belt may be 5 or 6 feet apart. These rollers are of several different types; they may be arranged to support the chains only as in Fig. 81[1], or the belt itself directly.

Sometimes the return half of the belt is not carried on rollers, but the edges of the plates rest simply on lengths of angle iron. In the same way the upper surface of the belt is partly supported by an angle iron running along either side; it is as well to use rollers in addition for this part of the belt, as the pressure of the loaded belt rubbing with its whole weight along the angle iron would cause considerable friction. Whether the frame, which carries the belt, be of wood or iron, it is advisable to have on either side of the belt an iron ledge 3 or 4 inches broad, upon which lumps of mineral may be broken with a hammer when necessary for better sorting. It is sometimes difficult to ensure that the mineral shall be distributed uniformly over the belt; this may be in great measure effected by means of a rake. One of the best forms consists of an iron bar furnished with teeth which is moved backwards and forwards across the belt near its top end. Such a rake with a to and fro travel of 3 or 4 inches, making 20 to 30 strokes per minute, is quite satisfactory; the system is however inapplicable when very large lumps of mineral have to be carried on the belt.

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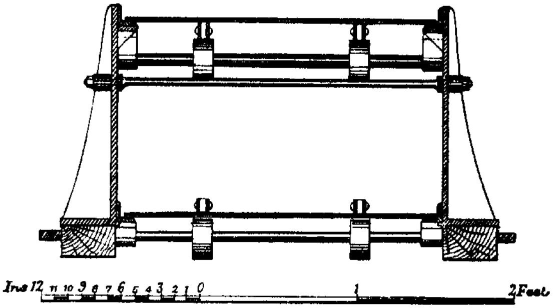
[1] *Trans. Fed. Inst. Min. Eng.*, "Improved Coal Screening and Cleaning," by T. E. Forster and H. Ayton, Vol. I. 1889–90, p. 83.



(*Scale for Figs.* 77 *to* 82.)

Fig. 81. Rollers for picking belt.

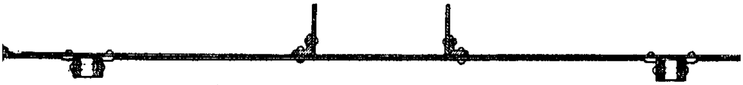


Fig. 82. Section of three-compartment belt.

Belts are often made with a central partition into which either the worthless stone picked out, or one particular class of mineral, when several kinds are sorted, may be thrown. It presents the same advantage as afforded by the upper shelf in the two-decker tables, namely of allowing the foreman in charge to inspect the materials thus picked out, and of carrying these away automatically, whereas in the other form they must be collected and removed by hand. This arrangement is illustrated by the section, Fig. 82, of such a belt, whilst a complete belt, as made by

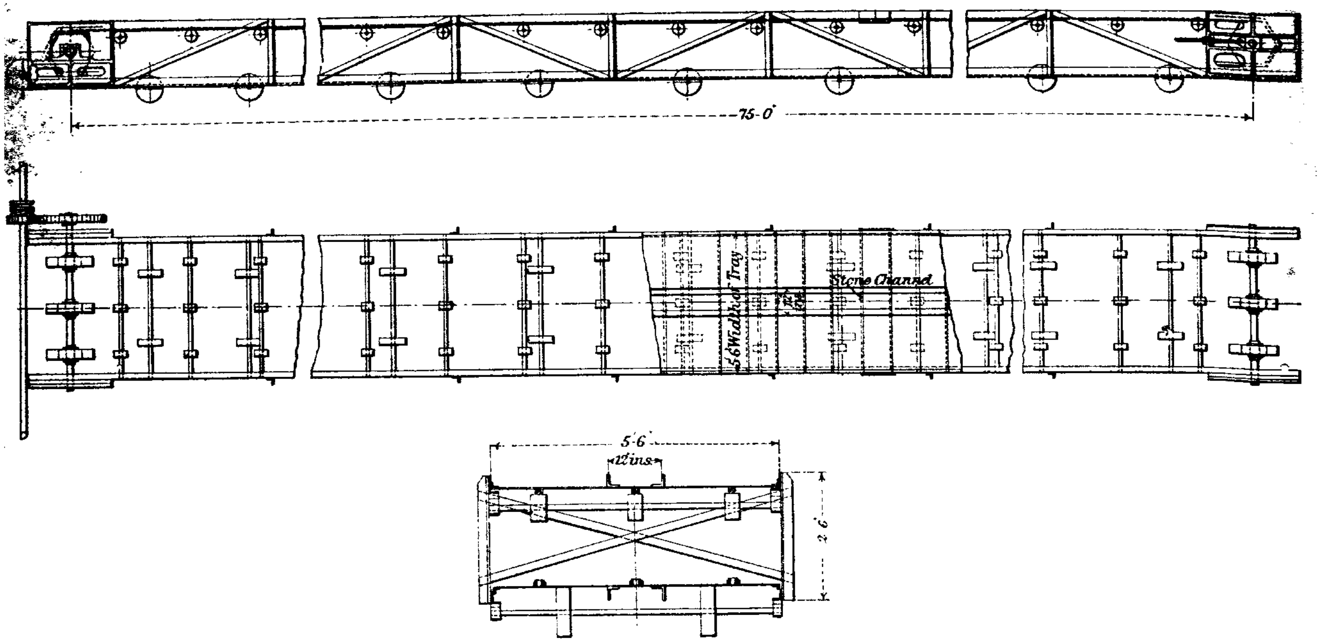


Fig. 83. Picking-belt with central compartment for stone. Plan, side and end elevations.

Messrs Joseph Cook, Sons and Co. Ltd., is shewn in Fig. 83. A belt with a central compartment may with advantage be wider than the ordinary form; 5 feet to 6 feet is a usual width, the central compartment

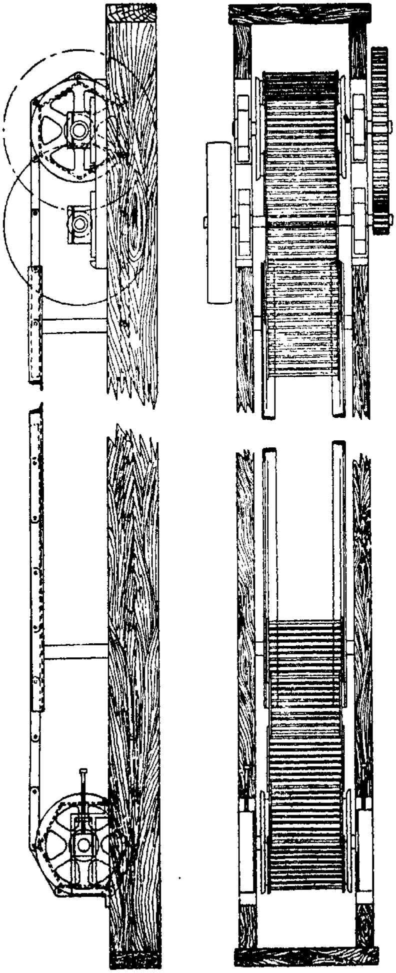


Fig. 84. Screening belt. Plan and elevation.

being 1 foot to 1 foot 6 inches wide as a rule. The delivery of mineral to the belt must be so arranged that none drops into this middle division;

this may be done by having two separate shoots, or one shoot with a central V (apex upwards) to direct the mineral right and left, or a distributing roller just below the shoot, the middle portion of which is of larger diameter than the rest and thus turns the mineral sideways. Such a belt 120 feet long, travelling at the rate of 100 feet per minute, is used for picking shale out of Cleveland ironstone, and can deal with 120 tons per hour, 25 per cent. being picked out.

Belts are sometimes arranged to screen the mineral that they are carrying; this is especially necessary when it has to be broken on the belt for the purpose of sorting. One of the best arrangements consists of cast steel plates about $\frac{5}{8}$ inch thick with oval slots of the required width cast in them; in this form the chains supporting the plates may be dispensed with, the plates hooking into each other at the ends, so that the plates themselves form as it were the links of a chain. Sometimes a belt of wire woven like locket work (see p. 17) is employed so as to screen the mineral; this arrangement is used to some extent for screening coals in the Lancashire coalfield. In Scotland belts made up of transverse bars (see Fig. 84) are used in the same way.

Belts have been made of various other materials, for example of lengths of flat rope sewn together, the separate ropes being kept in place laterally by strips of flat iron to which the various ropes are attached. Such ropes may be of hemp, aloes (used in Belgium), or steel wire. As a rule they are used at mines where flat ropes are employed for winding; when these ropes are too far worn for use in the shaft they can be utilised as picking belts. Except under these conditions, when their employment may be economical, the use of such rope belts has little in its favour. They are heavy, and wear out rapidly.

Canvas belts have been a good deal used of late years for picking purposes. The Robins Conveying Belt Company make a belt of canvas with a heavy covering of indiarubber, thickest in the middle, bent up by special rollers into a trough-like section. Such sorting belts have given quite satisfactory results in practice at several mines in Canada and America and elsewhere. A view of one of these belts is given in Fig. 85. It is usually made 32 inches wide, and will carry 28 to 50 tons per hour at speeds of 30 to 60 feet per minute. In a Norwegian mine similar belts are used for picking hard pyritic ore, and are found to wear well; such a belt 2 feet wide, 49 feet long, travelling at 49 feet per minute, with 4 to 5 pickers at work, deals with 20 tons of copper ore per hour, and is quite satisfactory.

When belts are used for coal picking it is important that the coal shall be discharged from the belt as smoothly and with as little breakage as possible. The drum at the discharge end should be kept as small as possible with that object, but in the most usual case of steel plate belts, care must also be taken that the plates are not made too narrow and the joints too numerous. With woven rope, canvas or indiarubber belts, where the drums may be round, a shoot can be placed close up to the drum and the coal allowed to slide quietly down it. Telescopic shoots, or shoots hung by chains and counterpoised, are often used; these can be lowered or raised to suit empty or full waggons and the breakage thus diminished. A well-known device for this purpose is the swinging jib end of Messrs Wood and Burnett, made by Messrs Coulson and Co., Ltd., and shewn in Fig. 86. In this arrangement the delivery end of the belt is carried on a hinged jib, suspended by chains and counterpoised so as to be easily raised or lowered as required. Another method is the swinging tray designed by Mr Chambers; this tray, Fig. 87, which may also take the form of a perforated plate and thus act as a screen at the same time, is moved by an eccentric driven off the machinery that drives the belt, so arranged that the upper lip of the tray is always kept close up to the edge of the discharging drum even though the latter be polygonal.

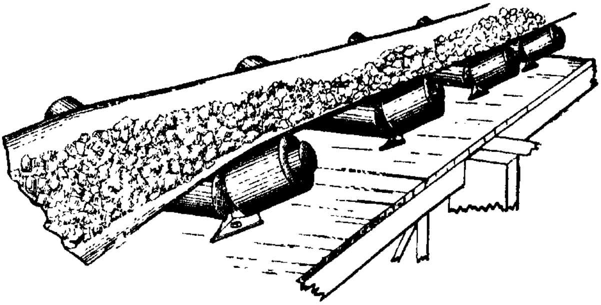


Fig. 85. Robins' indiarubber belt.

A very usual arrangement of a colliery heapstead comprises a number of picking belts each fed from a shaking screen, smaller conveyor belts running at right angles to the main belts and carrying off the small coal from below the screens, as also the stone and any inferior qualities of coal that may have to be picked out on the belts, and discharging each to separate waggons. The dimensions and capabilities of picking belts vary greatly, as shewn by the following table, in which

	1	2	3	4	5	6	7	8	9	10
Length of belt	60 ft.	70 ft.	48 ft.	70 ft.	66 ft.	54 ft.	57 ft.	110 ft.	60 ft.	100 ft.
Width of belt	4 ft.	4 ft.	4 ft.	4 ft.	4 ft. 8 in.	4 ft. 8 in.	4 ft.	4 ft.	4 ft. 6 in.	4 ft. 6 in.
Speed of travel per minute	60 ft.	70 ft.	66 ft.	50 ft.	26 ft.	58 ft.	45 ft.	38 ft.	40 ft.	50 ft.
Number of pickers	18	9	8	13	20	12	17	15	10	25
Distance apart of pickers	6·5 ft.	15 ft.	15 ft.	10 ft.	6·5 ft.	9 ft.	7 ft.	4·5 ft.	8 ft.	8 ft.
Coals treated per day, tons	348	295	245	280	370	420	100	102	300	500
Best coals got, tons	330	242	233	255	360	419	?	97·5	285	412
Inferior coal picked off, tons	5·6	—	4·5	17·3	—	—	—	—	—	—
Stones, band, pyrites, etc. picked off, tons	12·2	7·7	8·3	6·5	—	—	—	—	—	—
Total picked off, tons	17·8	7·7	12·8	23·8	10	9	?	4·5	15	87·5
” ” per cent.	5·10	2·67	5·20	8·50	2·70	2·14	?	4·41	5	17·5
Tons passed on to each belt per hour	35	29	25	28	37	42	10	10	30	50
Tons picked off each belt per hour	1·79	0·77	1·29	2·40	1·0	0·9	?	0·45	1·5	8·8
Tons treated per picker per hour	1·80	3·22	3·30	2·0	1·85	3·5	0·6	0·7	3·0	2·0
Tons picked off per picker per hour	0·097	0·086	0·172	0·178	0·05	0·075	?	0·03	0·15	0·35
Cost of wages per ton handled, pence	—	—	—	—	0·996	0·578	2·981	2·614	1·277	about 1·8

NOTES. Cost of lubricants per annum for No. 5 belt £2. 10*s*., for No. 6 belt £2. 5*s*., for No. 7 belt £3. 0*s*. Horse-power required a number of typical examples from the Northumberland and Durham coal-field are given. The power required to drive an ordinary picking belt varies with its size from 2½ to 5 H.P., and the cost of lubricants is usually between £2 and £3 per annum. The up-keep of a belt is

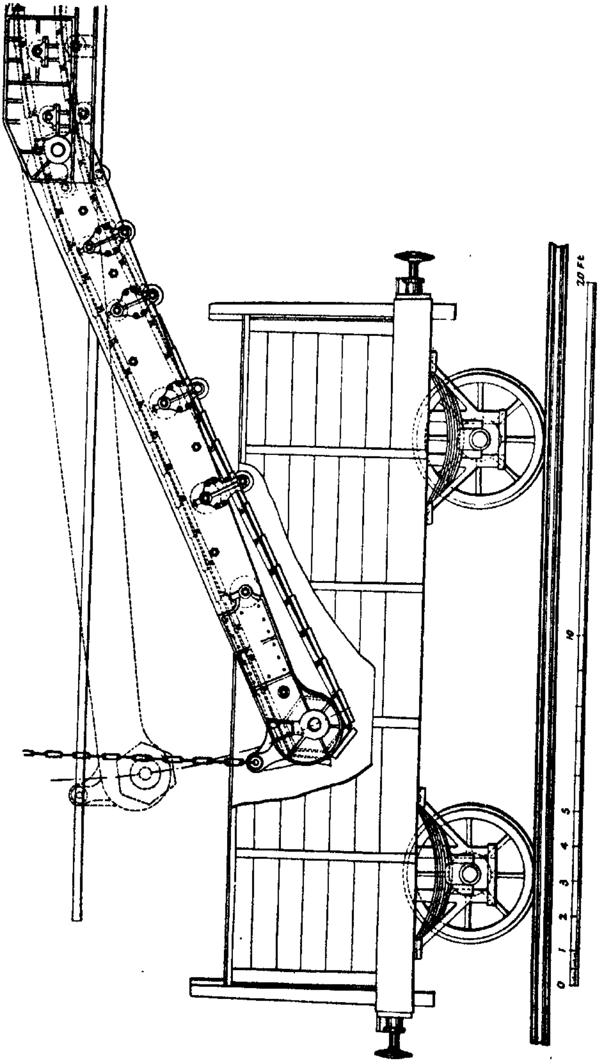


Fig. 86. Jib end of picking belt. Sectional elevation.

practically nil for the first few years, but soon becomes an important item when plates, links, etc., have to be renewed; it may be roughly averaged at £50 per annum. As will be seen from the table, the capacities of the various belts and the quantity of work performed by a picker vary within such very wide limits that it is scarcely possible to give any average figures.

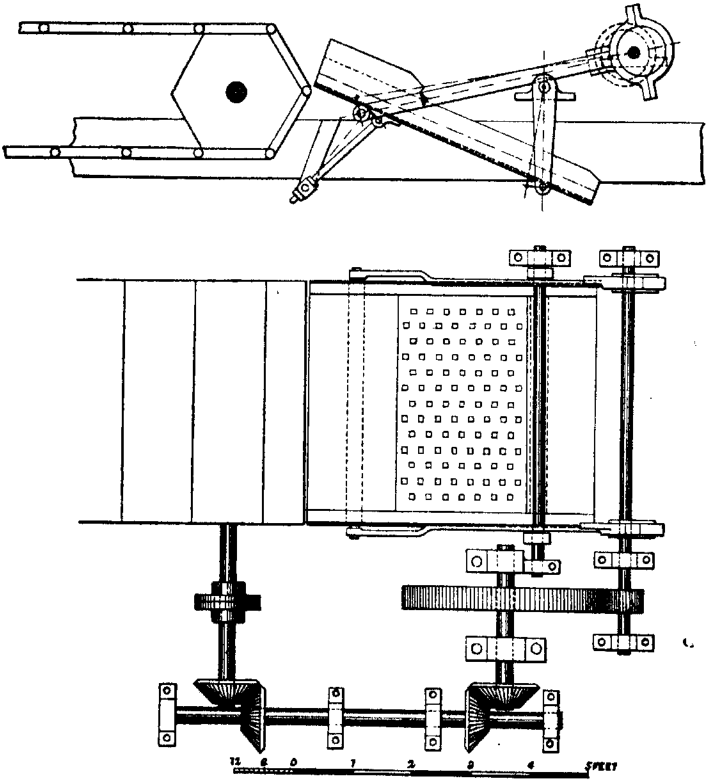


Fig. 87. Chambers' swinging tray. Elevation and plan.

Washing appliances. The word washing is here used in its more ordinary sense of cleaning or freeing from mud or clay, the French word "débourbage" exactly expressing its meaning. It is an operation that is often preliminary to sorting or to other dressing operations, but is sometimes the only process that a mineral needs to fit it for the market. This is very often the case with calamine and brown haematite, both of which ores often occur embedded in a tough tenacious clay, the presence of which greatly detracts from the commercial value of these ores; once this clay is removed, the ores need no further preparation. On the other hand many ores, notably galena, zinc blende, copper pyrites, gold ores (banket), etc., require washing to free them from adherent dust, clay, dirt, or powder smoke, in order to enable them to be properly sorted. As has already been seen this washing is frequently combined with screening and with sorting, the minerals being washed when lying on fixed or moving screens or upon sorting tables, and no special appliances being in such cases needed. It must not be forgotten that the washwater is apt to carry away much valuable ore in the form of fine slime, especially in the case of gold or silver ores; the water must therefore be led to settling pits in which the valuable slimes may be deposited and thus recovered.

Washing machinery as used for ores of the first-named class is mostly of one or other of two types; either fixed troughs in which the mineral is carried along by revolving paddles, or revolving drums.

Trough washers. These are known in America as "Logwashers," a convenient term that may well be adopted. In its simplest form this machine consists of a wooden trough lined with wrought or cast iron plates, set at a slight inclination to the horizontal. A shaft to which flat paddles are attached runs the full length of the trough and revolves slowly; the paddles are set spirally round the shaft and are arranged at an angle so as to gradually work the material up the trough. The material to be washed is delivered into the trough at its lower end and discharged at the upper; in its progress up the trough it meets with streams of water, which soften and wash off the clayey matter, which escapes at the lower end of the trough in suspension in water. These logwashers are mostly made double, a pair of shafts with paddles revolving side by side in one trough. Such a double logwasher, as made by the Allentown Foundry Company of Pennsylvania is shewn in Fig. 88.

The troughs may be made of wood, but wear out rapidly; cast iron segments bolted together answer well; cement has also been used. The capacity of the machine varies widely, as also does the water consumption, according to the character of the material being treated.

A group of four single washers are used at Longdale, Virginia[1], for washing brown haematite ores. Details of these are shewn in Figs. 397, 398, where the entire plant is described as a whole (p. 501). The troughs are wooden frames carrying cast iron plates, which fit together to form a semicircular trough 2 feet 6 inches in diameter and about 18 feet long, set at a grade of ¾ inch to the foot. The "logs" consist of cast iron pipes 17 feet 5½ inches long, 11½ inches outside diameter, and ¾ inch thick, to which are bolted the flat paddles, 8 inches long and 3½ inches broad; these are set at an angle of 25° to the axis of the shaft with a 5 foot pitch, there being 8 rows spaced equally round the pipe. The rate of revolution is 12 per minute; the washed ore leaving the upper end of the trough passes through a short conical trommel made of sheet steel with holes $\frac{3}{16}$ inch in diameter. Each washer treats about 50 tons of ore per day.

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. xxiv. 1894, p. 34.

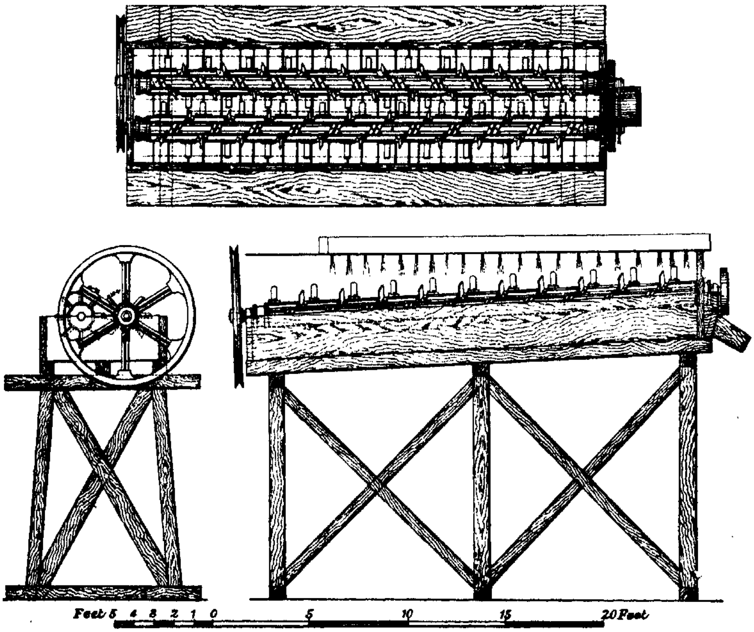

Fig. 88. Double log washer. Plan, side and end elevations.

Similar appliances are in use in various parts of the world; this type of washer is used, e.g., in various parts of Spain (where it is usually spoken of by its French name of "Patouillet") for washing iron and other ores.

Washing drums consist of cylindrical or conical drums made of boiler plate, and often of very large dimensions; as in the corresponding cases of trommels the first named are sometimes set with their axes slightly inclined, the latter always with their axes horizontal. This inclination of the axis is, however, by no means indispensable in the case of washing drums, in which the material to be washed is always propelled up the slope when there is one; the essential point is that the appliance should be so arranged as to possess an ample sump of water for soaking the ore; for this purpose the ends are partly closed with iron rings or very obtuse cones. The inner surface of the drums is provided with ribs, paddles, or teeth set spirally, by means of which the ore is carried forward and ultimately discharged.

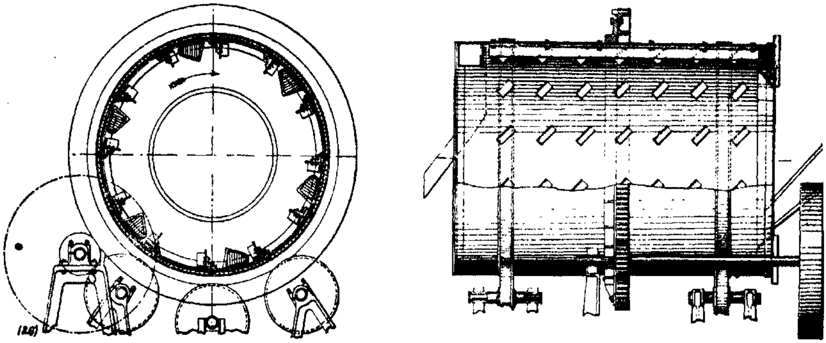

Fig. 89. Drum washer Vertical and transverse sections.

Fig. 89 shews a washing drum as manufactured by the Humboldt Engineering Company. It is supported on friction rollers and caused to revolve at a slow speed (8 to 12 revolutions per minute) by means of gearing. Both ends are partially closed by rings of iron plate so as to maintain a considerable depth of water in the bottom of the drum. The ore is delivered into the drum down a shoot and discharged at the opposite end. These drums are made up to 7 feet 6 inches in diameter and 18 feet in length. Such a drum 10 feet long and 5 feet in diameter can wash 17 to 25 tons of ore per hour, making about 7 revolutions, and using about 10 cubic feet of washwater per minute. It requires 4 or 5 H.P. to drive it, one-third of which is absorbed by the machine

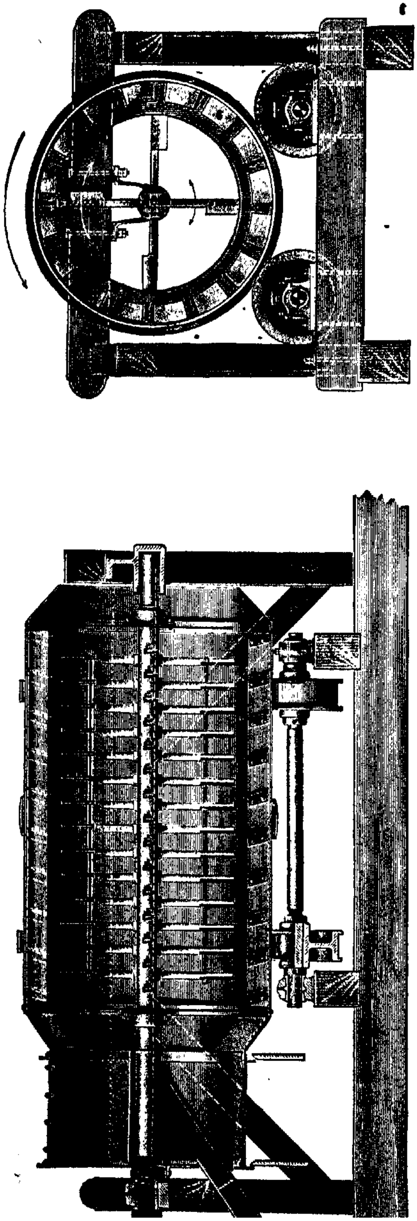

Fig. 90. Crickboom washer. Vertical and transverse sections.

running empty. Conical drums have a grade varying from 1 inch to 1¼ inch to the foot.

A similar washing drum is in use at Barrow[1]; it is 19 feet long, 4 feet 6 inches in diameter; to the inside a double spiral made of 3 inch angle iron is riveted, the spiral having a 20 inch pitch. Between the spirals there are iron teeth; it is set at a grade of 1 inch to the foot, is carried on friction rollers and driven by gearing at the rate of 9 revolutions per minute.

At Santander a cylindrical drum 13 feet long, by 6 feet 6 inches in diameter, terminating in a cone 5 feet long tapering to 1 foot 8 inches in diameter, and making 15 revolutions in 2 minutes, will wash about 400 tons of crude ore-clay in a 10 hours day, producing about 30 per cent. of washed ore, with a water consumption of nearly 1 ton of water per minute.

The Crickboom trommel, shewn in Fig. 90, is practically a combination of washing drum and logwasher, consisting of a cylindrical washing drum similar to those last described, in the axis of which is placed a shaft, furnished with arms or knives extending nearly to the walls of the cylinder. This inner shaft is rotated at a relatively high rate of speed in the opposite direction to the cylinder itself and pugs up the clayey matter adhering to the mineral, thus cleaning the latter very thoroughly; the material to be cleaned is advanced by teeth or ribs riveted spirally to the inside of the cylinder. These drums are made in various sizes ranging from 3 feet 3 inches to 6 feet 6 inches in diameter and from 5 feet to 10 feet 6 inches in length. The inner shaft is rotated at 15 to 20 times the speed of the cylinder. The following table shews, according to the Humboldt Engineering Company, the capacities of the various sizes of this machine when working on very clayey ores:

Drum		Revolutions per minute	Revolutions of shaft per minute	Horse-power required	Capacity tons per hour
Diam.	Length				
3′ 3″	5′ 3″	12	210—280	2·5	1
4′	6′ 6″	10	180—240	4	2
5′ 3″	8′ 3″—10′ 6″	8	150—200	7	4
6′	8′ 3″—10′ 6″	7·5	130—170	10	5
6′ 6″	10′	7	110—130	13	6

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[1] *Trans. Inst. Min. Eng.* Vol. XVII. 1898, p. 290.

The water consumption is usually 10 gallons per minute for each ton of ore washed per hour, but may rise to six times this amount. A medium-sized machine about 4 feet in diameter and 8 feet long weighs about 4½ tons and costs about £160. This drum (as is indeed the case with most washing drums) is usually supplemented by an ordinary screening trommel into which the material drops from the washing drum, and in which mud, sand, and slimes are removed.

CHAPTER IV.

COMMINUTION.

By comminution is meant generally the breaking down of mineral to smaller sizes; the words breaking, crushing, grinding, pulverising, etc. are used somewhat indiscriminately; it would no doubt be best to restrict the word breaking to coarse breaking, and crushing, etc., to finer breaking, but this practice is not at all generally followed. Mineral, as it comes from the mine, comes in pieces of all sizes, from lumps of as much at times as a couple of cubic feet or even more, down to small chips and dust, depending partly upon the nature of the mineral and partly upon the methods used for its extraction. It is generally considered best to get the mineral in the largest possible lumps, provided that these are not too large to be readily handled and transported; in some cases, e.g. steam-coal or iron ore, the value of the lumps may be two or three times that of the small stuff, but even when all the mineral has to be pulverised, it is considered better to get it out of the mine in fair-sized lumps, as these are less liable to loss during transport from the working face to the dressing plant; moreover lump ore can be hand-picked as a preliminary to further treatment, whilst fine ore cannot.

The principles upon which comminution depends are various. A piece of mineral may be broken down by:

1. **Crushing.** This implies exposure to a pressure which is greater than the resistance to crushing, or ultimate crushing strength, of the mineral; in this case more or less shearing stress is always developed together with the true crushing stress. It would seem as though the shearing of a body under compressive stress is due to the deformation of the mineral before it yields under the stress; the lower therefore the elasticity of the body, and the nearer its elastic limit approaches its ultimate resistance to crushing, the less is it likely to yield by shearing and the more by simple crushing. As a general rule both forms of yielding are set up concurrently in any mass of mineral under compression.

The maximum pressures which minerals can resist vary within very wide limits; as a rule the most difficult mineral substances to crush are the fine-grained comparatively homogeneous basic eruptive rocks.

The following table shews the pressures required to crush certain mineral substances:

Material	Ultimate strength in tons per square inch[1]	
Bituminous Coal ...	1·29 to 7·91	Average of 12 specimens 2·992
Limestone	0·72 to 1·65	Average 1·34
Marble	5·20 to 6·83	
Sandstone	0·95 to 4·88	
Slate	4·34 to 6·89	
Granite, Syenite ...	3·59 to 9·29	(Generally over 6 tons)
Basalt	7·11 to 10·69	
Porphyry	11·68	
Jasper	11·62	

2. **Shattering.** When a piece of mineral is struck sharply, the portion struck may be detached by the violence of the impact; whether any portion can be thus broken off depends upon that quality which is known as brittleness and also upon the nature of the blow. The momentum of the impinging mass plays a great part in determining whether or not the mineral struck will be shattered, but the question is not wholly one of momentum, because a small mass moving at a high velocity is less effective than a heavy mass moving more slowly, even though the momentum be the same in both cases. The mechanics of this everyday phenomenon are still very imperfectly understood, nor is it possible to frame any satisfactory definition of brittleness or to say upon what other properties it depends. Like hardness it is connected with the force with which the particles of the body cohere, but it is not altogether dependent upon hardness, some hard bodies being far more brittle than softer ones. Amongst mineral substances this may be well illustrated by quartzite and basalt; examples of these two rocks may be selected having very similar textures, when it will be found that the quartzite is far the harder of the two, but at the same time the more brittle, the magnesian minerals that enter so largely into the composition of these basic eruptive rocks being specially notable for their toughness. In mineral substances this quality is further complicated by their cleavage; a highly cleavable mineral will split easily under a blow that

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[1] The above table is compiled from experiments by Mallett, Michelot, and Bach; the experiments on coal were made by the author.

strikes it parallel to a plane of cleavage. The diamond, the hardest of known substances, is easily cleaved by a light blow in the proper direction, whilst bort (which is practically diamond less perfectly crystallised) is far less sensitive to percussion.

3. **Abrasion.** A mineral substance can be easily ground when it is either itself soft, if it is homogeneous in texture, or when the cementing material that holds the various particles together has feeble cementing power, if it is heterogeneous. In practice abrasion is rarely made use of, and only in special cases; the most convenient substance of which the working parts of grinding machinery can be made is either steel or chilled cast iron, and both of these have a hardness inferior to that of quartz; as this mineral enters largely into the composition of all mineral substances, such grinding surfaces would evidently suffer so greatly that grinding down in iron appliances a mineral containing much quartz is almost out of the question.

Whilst the above are the three principles upon which the comminution of mineral substances must depend, it is impossible to base any satisfactory classification of the forms of apparatus employed upon their respective modes of action. This depends partly upon the fact that any one piece of apparatus usually acts in several different ways; most of those that act by crushing have usually more or less percussive effect also, and even sometimes act by abrasion to some extent. For example, edge rollers act more or less in all three ways; there are machines, such as the stamp mill, that depend upon impact pure and simple, but on the other hand there are other machines, such as rolls, that in one form act chiefly by crushing, in another chiefly by percussion, and in a third chiefly by abrasion. It seems best therefore to abandon any attempt at a scientific classification based upon principles, and to retain the usual practical classification which refers to the objects of comminution. In this respect we may distinguish:

A. Coarse breaking—often a preliminary operation.

B. Medium crushing.

C. Fine crushing (pulverising).

Coarse breaking may be further subdivided into hand- and machine-breaking; in groups B and C machinery alone is employed in practice.

A. COARSE BREAKING.

Coarse breaking is employed, either as a preliminary to further crushing, with or without sorting or sizing, or it may be the sole preparation for the market that is needed. When large quantities, or a particularly resisting mineral have to be treated, machine-breaking is indicated, especially where power is cheap, where skilled labour is not abnormally dear compared with unskilled, where materials for renewals, lubricants and other stores are not excessively dear, and where hand-sorting is not needed. Under the opposite conditions hand-breaking may be the more advantageous. Hand-breaking is often combined with sorting; it may indeed be fairly said that the latter is impossible unless combined to a certain extent with the former.

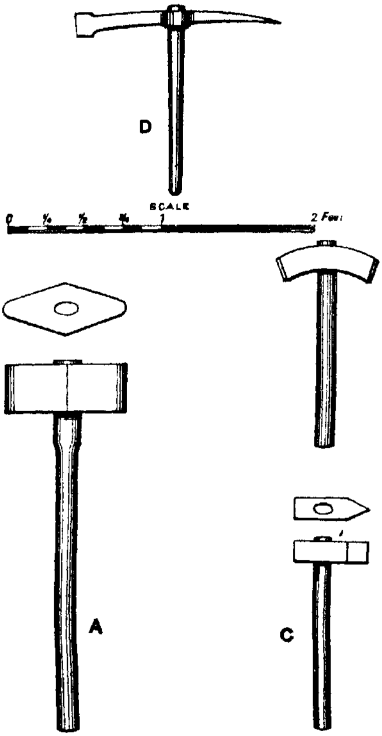

Fig. 91. Breaking hammers.

Hand-breaking. For breaking up large rocks a sledge-hammer is used; this usually weighs from 19 to 20 lbs., going at times up to 30 lbs. An ordinary sledge-hammer with well rounded or egg-ended panes may be used; a more frequent shape is that shewn in Fig. 91 *A*.

The operation of breaking with sledge-hammers is known in Cornwall as "ragging."

For smaller breaking, called "spalling" in Cornwall, lighter hammers are used. These are either small double egg-ended hammers weighing 2 to 4 lbs., attached to a long flexible handle, or else are rather heavier, 3 to 6 lbs., with a shorter handle and of the shape shewn in Fig. 91 *B*; these are used in one hand, this being known as "cobbing" in Cornwall. When sorting is a special object one pane of the hammer is often chisel-shaped, as in Fig. 91 *C*. In Cornwall hand-crushing used to be carried still further, and is still to some small extent, a single flat-paned hammer being used known as a "bucking-iron"; such crushing is now however mostly performed by machinery. For both cobbing and bucking the mineral should be laid on a thick iron plate.

Lumps of coal are cleaved, in order to separate pieces of different qualities, or to remove band or shale, by means of a light single-hand pick, about 15 inches long (Fig. 91 *D*), called a 'snap" in the north of England.

Ordinary breaking is usually performed on breaking floors, arranged exactly like picking floors (see p. 78). They are usually paved with strong stones. Cobbing and bucking are generally carried on in special sheds, which must be well lit so as to enable the workers to sort the minerals properly. When breaking and picking are carried on together, a gang of from 10 to 20 workers will be needed to break 100 tons of ordinary quartzose ore down to say 1½ inch ring and to sort it ready for fine crushing. In the case of complex ores, where a number of different classes have to be made, a relatively larger staff will of course be required. The cost of breaking by hand and picking may range from 4*d.* to 1*s.* per ton, inclusive of all expenses. When breaking alone has to be done with little or no picking, each worker may be reckoned to break from 20 to 60 cubic feet of mineral per day, according to its character. A good workman can break 1½ cubic yards of hard basaltic rock or 2 cubic yards of compact limestone to road-metal size (to pass through a 2 inch ring) in a day of 8 hours. The wear and tear of the breaking hammer amounts to about 1½*d.* per cubic yard of basalt, and 1*d.* per cubic yard of limestone. In breaking and sorting hard pyritic ore at Rio Tinto, an average day's work is 4 tons per man.

Machine-breaking. There are two types of mechanical rock-breakers, the **reciprocating** and the **rotating**, the former being represented typically by the Blake and the latter by the Gates rock-breaker (or ore-crusher as it is often called in America).

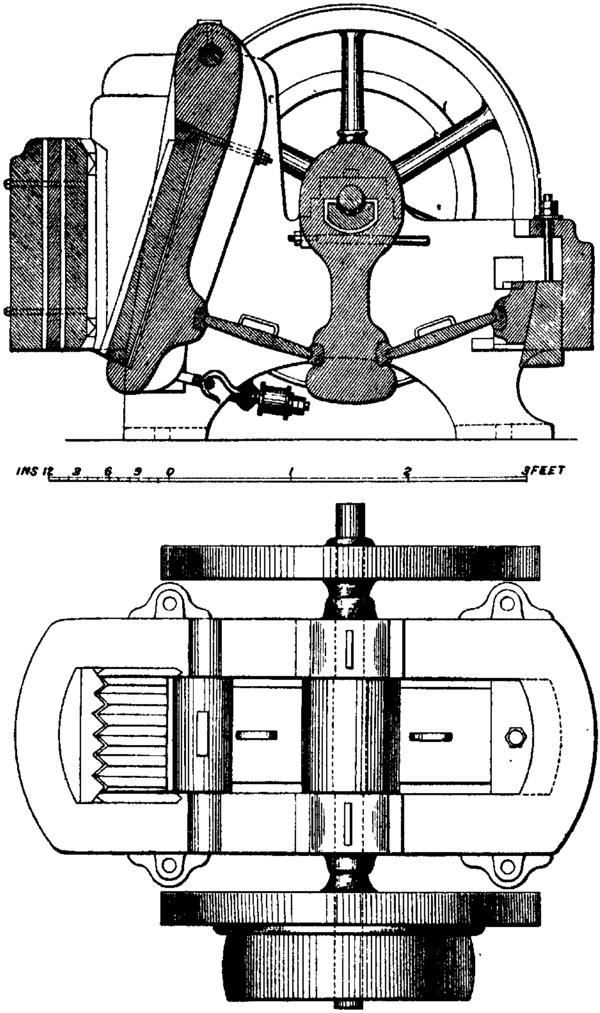

Fig. 92. Blake rock-breaker. Plan and vertical section.

Reciprocating breakers may be divided into three classes; (a) those in which the motion of the jaw is greatest at the bottom (b) those in which it is greatest at the top, and (c) those in which it is uniform all the way down, these three classes being represented typically by the Blake, the Dodge, and the Forster Rock-breakers respectively.

Class A. *Jaw motion greatest at the bottom.*

Blake rock-breaker. A usual form of this machine is shewn in plan and section in Fig. 92, and in perspective in Fig. 93. The machine consists of a massive iron frame, in the front part of which is fixed a plate of steel or cast iron against which the mineral is broken, and which is usually known as the fixed jaw. The iron frame carries a horizontal shaft with driving pulleys and a pair of heavy fly-wheels; on this shaft is forged a massive eccentric, running nearly the full length of the inside of the frame. This eccentric actuates an eccentric plate or pitman, which carries bearings into which fit two toggle plates. The rear toggle plate rests in a bearing, carried on a folding wedge. The front toggle plate bears against the swinging jaw, which is a heavy plate of iron hung on a shaft in such a way that it can vibrate backwards and forwards. Like the fixed jaw, the swinging jaw is provided with a wearing face of steel or chilled cast iron. The swinging jaw is drawn back and held firmly against the toggle plates by means of a tension rod bearing against a solid indiarubber block that acts as a spring. Spiral steel springs are preferred by some makers, but are liable to break suddenly in use. The minimum

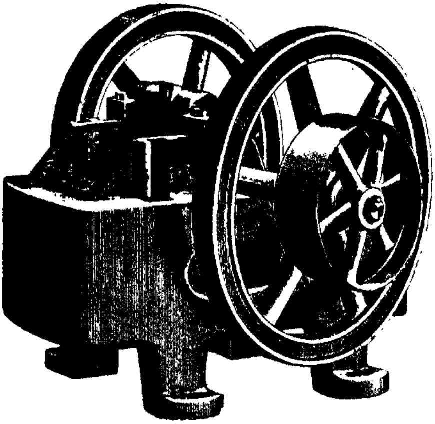

Fig. 93. Blake rock-breaker. Perspective view.

distance between the bottom ends of the fixed and swinging jaws determines the size to which the finished product is broken. This width of aperture can be adjusted within certain limits by drawing up or letting down the folding wedge by means of the bolts that hold it in place. The capacity of the machine is controlled by the area of the opening between the upper ends of the jaws, and the capacity of the machine is in ordinary practice indicated by dimensions of this opening. The sides of this opening are generally protected by a couple of steel liner plates or cheek pieces, which in some makes also keep the fixed jaw in position.

The action of the machine is simple: the wedge-shaped space between the fixed and swinging jaws being filled with the material to be broken, when the eccentric shaft is revolved, the swinging jaw is pressed forward by the action of the toggle joint; it is well known that by the construction of this particular form of joint the pressure exerted at the end of the free toggle is a gradually increasing one, and finally becomes enormous, although exerted only through a very small space. This is precisely the action best calculated to split up the material between the jaws, and as the swinging jaw is drawn back, the material crushed small enough to pass between the lower edges of the jaws drops out, the remainder of the material settling down a little; at the next forward swing the same crushing action is repeated, and so on until all the material between the jaws has been crushed and has dropped through. In practice of course the material to be crushed is fed in continuously at the upper end, so that crushing and discharge of crushed material proceed continuously.

The degree of crushing is governed by the ratio between the width of the top and bottom jaw apertures, or for a given length of jaw by the angle between the two jaws, which is usually 20° to 25°. The larger it is, the greater the crushing efficiency of the machine; it must not however be too large, or there is risk of pieces of mineral being projected upwards out of the jaws. In Fig. 94 let KL and MN be the fixed and swinging jaws respectively, inclined to each other at an angle θ; let C be the centre of a particle of mineral between the jaws touching these in the points A and B. The particle is then subject to the forces P the pressure due to the swinging jaw, R the resistance of the fixed jaw, and W its own weight. The force P may be resolved into the force $P \cos \phi$ acting along the line BA and $P \sin \phi$ acting in the direction S, at right angles to BA, ϕ being the angle between S and MN; ϕ is evidently by construction $= \frac{1}{2}\theta$. As P is always very great compared to W, the magnitude of the force is never in question because $P \sin \phi$ will always be so much greater than W that the latter may be neglected, hence the direction alone of the forces has to be considered, and if $\phi > \tan^{-1} \mu$, where μ is the angle of static friction, it is evident that the particle will be forced upwards. Hence we get for the safe construction of the appliance the condition that $\theta < 2 \tan^{-1} \mu$, this being the condition that has to be fulfilled in order to prevent pieces of stone from being projected upwards.

It is evident that, according to the length of the pitman for any given throw of eccentric, the toggle plates may be in a straight line, and therefore the crushing pressure a maximum, when the eccentric is at the top, or at bottom of its stroke, or at any intermediate position. In the two former cases the jaws will only be forced together once for each complete revolution of the shaft, in the latter twice, so that by this last arrangement, the number of vibrations being twice as great, the working capacity of the machine is increased. This is the usual arrangement in practice, the only objection to it being that the pitman is alternately in compression and in extension at the moment that the maximum pressure is being exerted, and that this double strain must be provided for by suitably strengthening the pitman. The swinging jaw should be so suspended that the centre of the shaft carrying it should lie in the plane of the working face, or slightly in

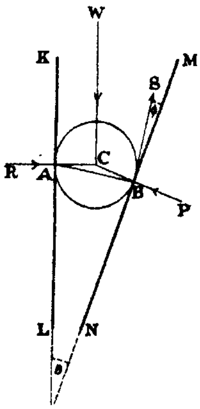

Fig. 94. Diagram of action of rock-breaker.

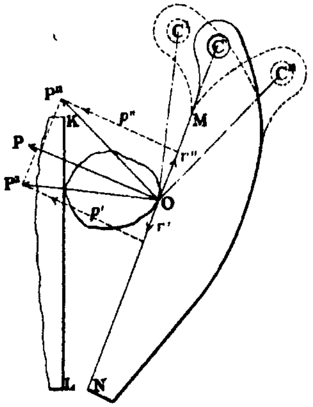

Fig. 95. Diagram of rock-breaker jaws.

front of it, but never behind it. Thus in Fig. 95 let *MN* and *KL* be the working faces of the swinging and the fixed jaws respectively; if the centre of suspension of the swinging jaw is at *C*, a point lying in the prolongation of the line *NM*, it is evident that each point in that line is exerting a pressure perpendicular to that of the face, the direction of pressure *P* at any point such as *O* being tangential to the circle described with centre *C*, of which *CO* is a radius. If the axis of the shaft *C′* be in front of the line *MN* it is evident that the line of pressure *P′* is inclined to the face and may be resolved into *p′* perpendicular to it and *r′* parallel to it, the latter tending to press the point *O* downwards, and therefore helping the crushing action to some extent. If however the axis lie behind the face *MN* as at *C″*, the pressure *P″* exerted is resolved into *p″* the pressure at right angles, and *r″* the pressure parallel to the jaw; the latter portion tending to force the point *O* upwards and therefore out of the machine. To avoid any such danger, the centre *C* is usually kept a little in front of the face *MN*. The greatest amount of wear comes upon the lower edge of the jaws, so that after the jaw has been in work for some time the plane of the jaw becomes tilted backwards, so to speak, on account of this wear, and unless the axis of suspension were kept well in front of the plane when new, it might come to lie behind it when worn.

Details of construction of a modern pattern are further shewn in the sectional perspective view, Fig. 96, which represents a breaker, the body of which is made of cast steel, as made by Hadfield's Steel Foundry Co. Ltd.

Variations in construction are numerous but not very important. The frame is sometimes built up of steel plates held together by strong tie bolts, for the sake of portability; this construction should be avoided wherever at all possible. The dead weight of the frame is a positive advantage to the machine, tending to ensure smooth working and to confine the wear to the jaws. The force that crushes the mineral is evidently resisted by stresses tending to burst the end plates apart, and these are very severe, and it is difficult to get anything except an extremely massive casting to resist them. It is on this account that Hadfield and Jack employ cast steel instead of cast iron in the machine shewn in Fig. 96, so as better to resist the severe tensional strains set up in the frame.

The removable faces of the jaws are secured in various ways—by means of T-headed bolts sliding in slots in the back of the jaws, by means of conical-headed bolts through the jaws, by running the jaws with lead or zinc, etc. The first is perhaps the best method; whichever is adopted, care should be taken that the jaws are firmly and evenly bedded.

The jaws are sometimes, but rarely, made of steel plate, which is only suitable to the crushing of soft material.

"Pin plates," consisting of plates of ordinary mild steel, drilled with a number of holes 1 to 2 inches in diameter, into which plugs made of short pieces of round tool steel are forced, have also been used.

For breaking the harder minerals cast steel or chilled cast iron is preferable; the latter is often selected when the machine is situated within easy access of an iron foundry; otherwise cast steel is perhaps the best material, and where the cost of transport is a serious item, chrome steel and, above all, manganese steel may be recommended. The faces of the jaws may be either smooth or corrugated; in the latter case the corrugations are V-shaped in cross-section and run vertically, as in Fig. 96. The corrugated jaw is best suited to very coarse breaking only.

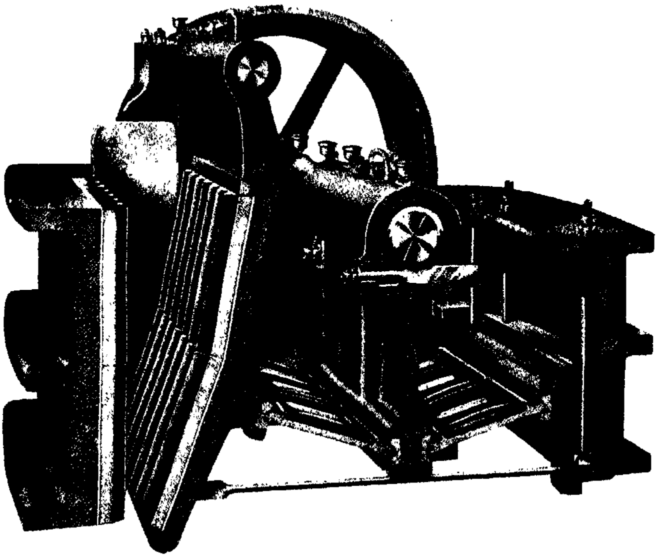

Fig. 96. Hadfield and Jack's rock-breaker. Perspective section.

The wear of the jaws varies within wide limits; it may be averaged at about 0·1 lb. of metal per ton of hard quartzose ore broken; it is often however a good deal less.

Blake rock-breakers are made of many different sizes. The following table shews the approximate sizes and capacities that are generally obtainable, but the practice of different makers varies somewhat:

Size of mouth	Tons of hard rock broken per hour	Weight of machine in tons	I. H. P. needed to drive	Revolutions per minute
10″ × 4″	2—3	2½	9	300
10″ × 8″	4—5	4	12	300
15″ × 9″	6—7	7	16	250—300
18″ × 10″	8—9	9	20	250—300
20″ × 12″	9—12	11	25	250—300
24″ × 18″	12—16	16	30	200—250
24″ × 24″	18—22	18	40	200—250
30″ × 24″	16—24	21	45	200—250

The above capacities are for breaking to about a 2 inch ring.

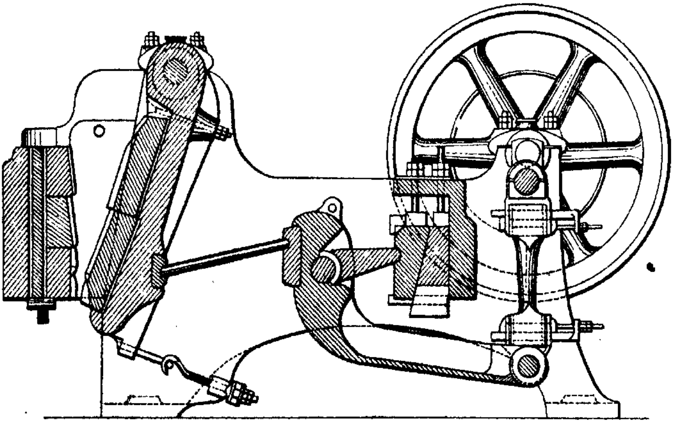

Fig. 97. Blake-Marsden rock-breaker.

A 15 inch × 9 inch machine costs about £150 and a 30 inch × 24 inch machine about £350.

The Blake-Marsden Lever Motion Stone-Breaker is shewn in section in Fig. 97. It will be seen that this machine, though similar in general principle, differs in that the lever that actuates the toggle plates is bent and is worked from a crank shaft by means of a connecting rod, instead of being actuated directly by an eccentric. This allows the driving shaft and its bearings to be kept further away from the vibrating jaws, a position that presents a good many advantages. The machine is however a little heavier and more expensive than the ordinary form.

A rather aberrant type, best considered here, is the Schranz "Sectorator[1]," shewn in Fig. 98.

In this machine the swinging jaw is curved and is suspended from a link in such a way as to have a rolling motion against the fixed jaw.

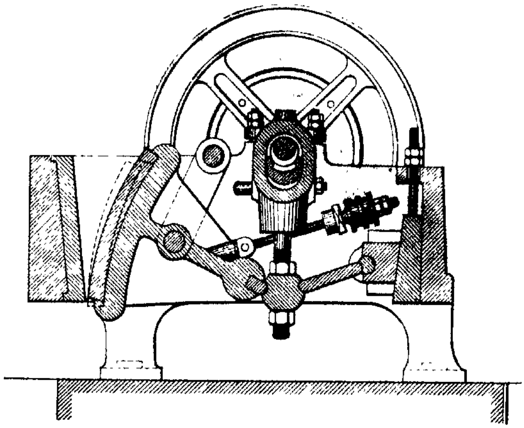

Fig. 98. Schranz Sectorator.

It is stated to break 2¼ tons per hour, of pieces varying up to 4 inch cube down to nut size with 200 revolutions per minute, and is said to consume no more power than an ordinary rock-breaker of equal dimensions. Its object is however to break down to about 0·3 inch, and hence it might fairly be classed among medium breakers rather than coarse breakers.

Rock-breakers have also been made in which both jaws vibrate, and others in which one swinging jaw vibrates between two fixed ones, so as to crush during both portions of the revolution of the eccentric shaft, and again with two swinging jaws and one fixed one, but none of these modifications have found any extended practical acceptance.

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[1] *Zeitsch. f. B. H. u. Sal. Wes.* Vol. xxxv. 1887, p. 263, D. R. P. 30477.

CLASS B. *Jaw motion greatest at the top.*

The Dodge rock-breaker is shewn in Fig. 99 in vertical section, plan and front elevation. This machine is somewhat different in

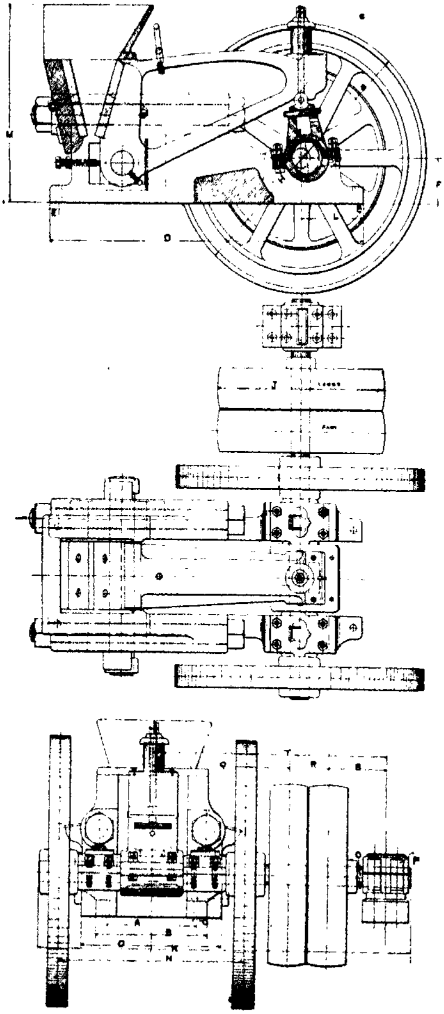

Fig. 99. Dodge rock-breaker. Vertical section, plan and front elevation

principle to the last-named. The flywheel shaft carries a strong eccentric which actuates the pitman, the lower end of which is connected with the longer arm of a massive bell-crank lever; the shorter arm of this lever carries the vibrating jaw. This jaw is therefore pivoted at its lower end, so that the width of discharge does not vary throughout the stroke. This width is capable of exact adjustment by means of the set screws shewn in the front of the machine, which move the bearings in which the swinging jaw is carried and tightens them firmly against packing plates. This machine gives accordingly a more uniform product than the Blake, but its capacity is less. It works well on hard material; owing to the mode of suspension of the jaw, the pressure exerted is greater the further the material to be broken descends between the jaws, until it is broken small enough to escape. The following table shews some of the more usual sizes of this machine:

Size of mouth	Tons of hard rock broken per hour	Weight of machine in tons	I. H. P. needed to drive	Revolutions per minute
9″ × 7″	1½	2	8	285
12″ × 8″	3	3	12	220
16″ × 10″	6	5	20	200

The above capacities are for breaking to about 1½ inch ring. The cost of a 12 inch × 8 inch machine is about £110.

The Bartsch rock-breaker, made by the Humboldt Engineering Co., is practically identical with the Dodge machine.

A rock-breaker known as the **Booth Combination Rock-breaker** is made by the Risdon Iron Works; it consists practically of a Blake set above a Dodge breaker, each of these having its own vibrating jaw, whilst the vertical fixed jaw is continuous. The wearing faces of the jaws consist of alternate flat bars of iron and steel set horizontally. The makers state that a machine having a mouth 9 inches × 15 inches will break 10 tons per hour to a 1½ inch ring and will require 8 H.P. to drive it; the machine weighs 4¼ tons and costs £200.

CLASS C. *Jaw motion the same at top and bottom.*

The **Forster rock-breaker,** Fig. 100, was at one time largely used, but is now comparatively rarely seen; it is not unlike the Dodge breaker except that the jaw vibrates about a vertical instead of about horizontal axis. The makers, Messrs Fraser and Chalmers, Ltd., stat that a machine having a mouth 7 inches × 18 inches will break 6 tons pe hour to about a 2 inch ring, using 5 H.P. and running at 300 revolutions per minute. The machine weighs nearly 3 tons and costs about £150.

In all reciprocating breakers a heavy flywheel is an essential part of the machine; the momentum stored up in this during that part of the stroke in which no breaking is done is available for overcoming the resistance to crushing of the material between the jaws in the other part of the stroke; even so, however, the action of the machine and the power it absorbs are very irregular and variable.

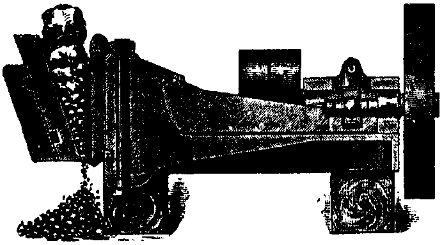

Fig. 100. Forster rock-breaker. Sectional elevation.

Rotating or Gyrating rock-breakers. All of these, which are very similar in construction, consist practically of a circular hopper in the shape of an inverted very acute-angled truncated cone, which is furnished with a conical steel liner which is the equivalent of the fixed jaw of the reciprocating rock-breaker; within this is supported a steel cone apex upwards, which is supported near its apex, whilst the lower end of its axis is carried round in a small circle by means of a crank, this gyrating cone corresponding to the swinging jaw. As the cone is carried round, any line on the generatrix will uniformly approach and recede from the inner surface of the cone within which it gyrates; material dropped into the outer cone will therefore be crushed between the walls of this fixed cone and the inner moving one. The angle of the two cones is usually made the same, and should in that case not exceed $\tan^{-1}\mu$, where μ is the coefficient of friction between the material to be broken and the steel breaking surface; or in other words the angle between the surfaces of the two cones must be less than $2\tan^{-1}\mu$ as in the case of the reciprocating breaker.

The Gates Breaker. The construction of this is shewn in section in Fig. 101 and in perspective in Fig. 102. It will be seen that the fixed annular jaw is supported in a massive casting, which is bolted to another casting that carries not only the body of the machine but also the bearing for the short horizontal driving shaft and the step for the vertical shaft by which the head is caused to gyrate. The steel shell against which the crushing takes place drops into the body and is held in place by another casting bolted to the top of the body, which at the same time forms a hopper, and which also carries the upper bearing of the vertical shaft. This bearing is steadied by three, or in the newer patterns, by two massive arms which divide the mouth of the machine into three (or two) openings, and thus limit the size of the pieces that each machine will take. The steel crushing head slips over the vertical shaft, the lower end of which is carried round by the eccentric driven by the bevel gearing as shewn. This head is not keyed to the shaft, so that it does not rotate with the latter, but receives a gyratory motion that brings it close to all parts in turn of the conical steel shell, without any rubbing or grinding motion being possible of the crushing surfaces or of any pieces of mineral lying between them against either surface.

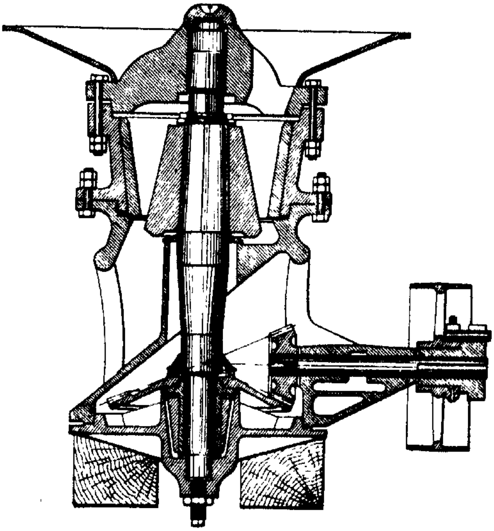

Fig. 101. Gates rock-breaker. Vertical section.

Though hardly as well suited as the reciprocating rock-breaker to small outputs, this machine is decidedly to be preferred when large quantities of material have to be broken ; under these latter conditions it is more economical of power and above all runs with far less vibration than the former type. A reciprocating crusher necessarily strains its foundations

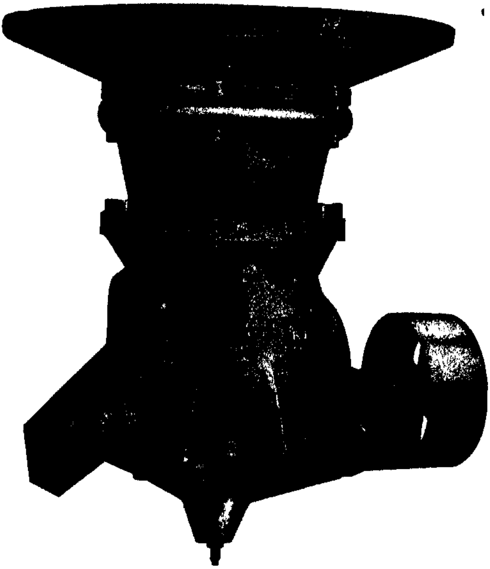

Fig. 102. Gates rock-breaker. Perspective view.

nore than does a rotating one ; the cost of renewals is however rather reater with the last-named.

The following table shews the approximate capacities of these nachines, according to the makers :

Diameter of hopper	Dimensions of each of (2) openings of mouth	Tons of hard rock broken per 24 hours	Weight of machine in tons	I.H.P needed to drive	Revolutions of driving shaft per minute
37½″	5″×18″	3½	2½	16	475
39½″	6″×21″	5	3½	20	450
44½″	7″×22″	8	6	30	425
51″	8″×27″	12	9½	40	400
59″	10″×30″	20	13	50	375
66″	11″×36″	25	18	60	350
120″	14″×45″	35	27¼	80	350
132″	18′×68″	85	40	100	350

The cost of a 10 inch × 30 inch machine is about £500.

The Comet rock-breaker made by Messrs Fraser & Chalmers, Ltd., is very similar, and other machines, like that made by the Union Iron Works, differ mainly in that the gyrating shaft is suspended from the top bearing instead of being supported by a step at the bottom.

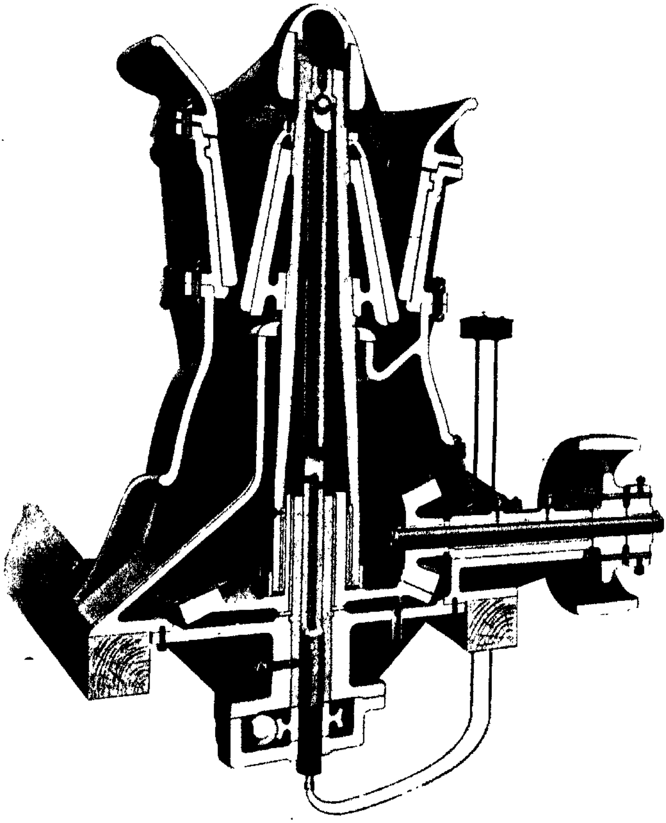

Fig. 103. Hecla rock-breaker. Perspective section.

A similar arrangement is adopted for the **Hecla** breaker made by Hadfield's Steel Foundry Co., Ltd., shewn in Fig. 103 in sectional perspective. In this pattern the spindle that carries the gyrating cone is hollow and is supported on a ball bearing carried on a vertical column inside the spindle; the whole of the breaker, except the foundation plates, is made of cast steel, and special attention is paid to the lubrication of the moving parts. The driving pulley is not keyed to the counter shaft, but is secured to a strong collar keyed on the shaft, by breaking pins, arranged so as to shear in case of any unduly hard material finding its way into the breaker.

In all forms the width of the discharge aperture can be altered by raising or lowering the vertical shaft that carries the gyrating cone; the amount of adjustment that is thus possible is somewhat limited.

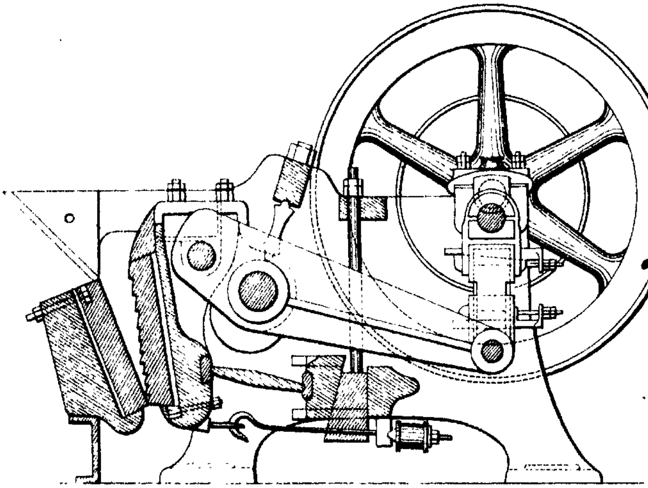

Fig. 104. Marsden's pulveriser. Vertical section.

The action of these machines is smoother and more uniform than that of reciprocating breakers, and as the breaking action is continuous a heavy flywheel is not required.

B. MEDIUM CRUSHING.

This term may conveniently be applied to breaking performed in machines which receive the mineral after it has undergone a first breaking, and which deliver it in the form of a coarse powder, the upper limit of which may be fixed at particles of about ¼ inch in diameter; of course a considerable amount is reduced to fine powder at the same operation. These machines work either by impact or by pressure; they are most typically represented by the well-known Cornish rolls which act in the main by pressure.

Marsdens' Pulveriser, Fig. 104, is somewhat similar to the Schranz Sectorator in mode of action, the swinging jaw also receiving a rubbing motion. In order to enable it to crush fine enough it is furnished with horizontal ribs or teeth like those of a file; when these teeth are worn away its efficiency is considerably diminished, but the machines do not seem to have come into at all general use, so that it is difficult to give any data as to the results practically obtained by them. The largest size machine made has a jaw aperture of 20 inches × 3 inches and weighs 6 tons; it is said to be able to crush 21 cwt. per hour to a

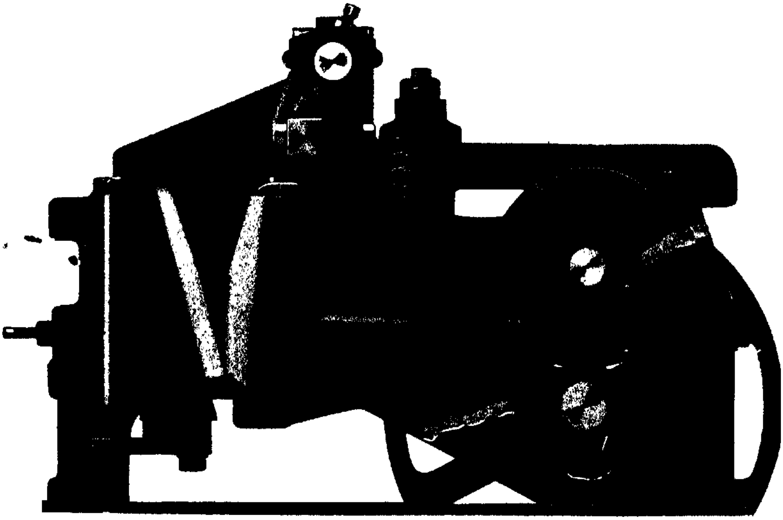
Fig. 105. Sturtevant roll-jaw crusher. Vertical section.

mesh of 30 holes to the linear inch, consuming about 20 H.P., the pulley speed being 300 revolutions per minute. This machine costs £250. The makers state that the capacity of the machine varies as follows according to the degree of fineness of crushing:

Holes per linear inch	Ratio of material crushed
30	100
20	110
16	120
10	145

The Sturtevant Roll Jaw fine crusher, Fig. 105, is also somewha similar in its mode of action.

A Fine Crusher is also built by the Gates Iron Works, similar i general construction to their rock-breaker. The concave die is howeve

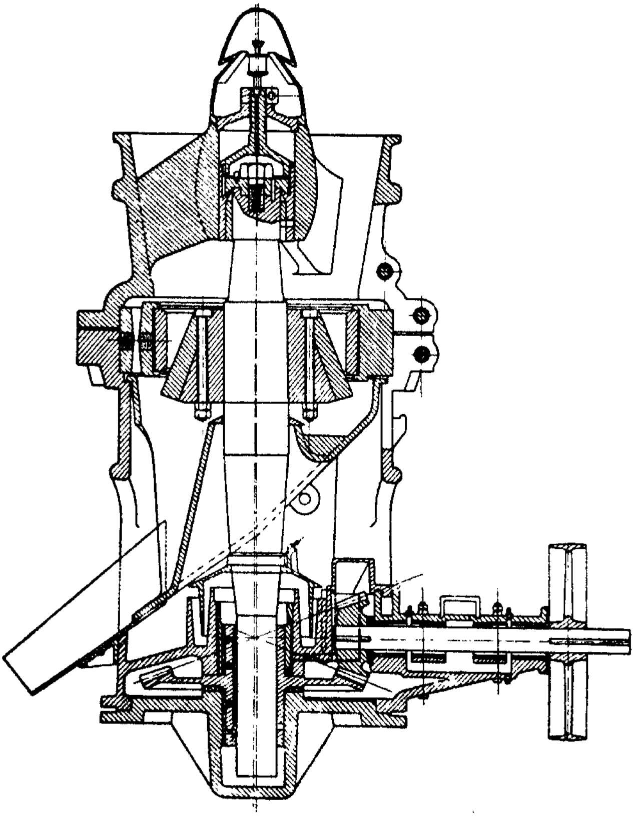

Fig. 106. Gates fine crusher. Vertical section.

here replaced by a cylindrical casting as shewn in the section, Fig. 106 and the conical head is much shorter than in the rock-breaker, and i suspended instead of being carried on a step. This machine is intende to take rock broken to a 3 inch ring and to crush it down to ½ inc mesh. Its capacity is stated to be 30 to 40 cwts. per hour and it consumption of power equal to about 15 H.P. The driving pulley is 24 inches in diameter and makes 600 revolutions per minute. The machine weighs 85 cwts. and costs £250. It has not yet come into general use.

Rolls. Although these machines, the most generally used of all forms of medium crushers, vary considerably in the details of their construction, the mechanical principles involved are everywhere the same. If two metal rolls be made to revolve so that the upper surfaces are moving towards each other as indicated in the diagram, Fig. 107, and a piece of mineral be placed between these surfaces, the mineral will be drawn in between the approaching surfaces by the friction between it and these surfaces (if the necessary conditions are duly observed) and will thus be crushed until the comminuted particles drop through between the rolls, which are not supposed to be in actual contact.

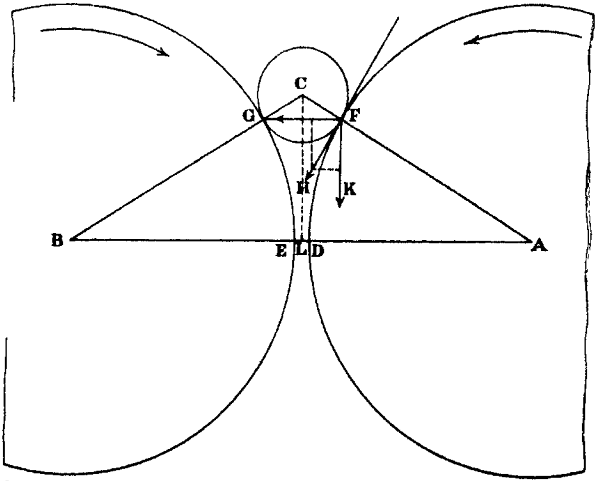

Fig. 107. Diagram of action of rolls.

It is necessary that the size of the particle to be crushed, and the diameter of the rolls, shall bear a definite proportion to each other, the size to which the mineral is to be crushed having also to be taken into account.

In Fig. 107 let *A* and *B* be the centres of the two rolls, and let *C* be the centre of the piece of mineral to be crushed (supposed spherical), the rolls touching this sphere in the points *F* and *G* respectively. Considering the roll *A* alone, the tangential direction of motion of the roll, *FH*, at the point *F* may be resolved in the directions *FG*, *FK*; the component *FG* of the force applied at the periphery of the roll tends to crush the sphere of mineral, the component *FK* to draw it downwards. In order that it shall be drawn downwards it is necessary that the limiting value of the angle *HFK* shall be $\tan^{-1}\mu$ where μ is the coefficient of friction between the surfaces (in practice $\mu = 0{\cdot}3$ and $\tan^{-1}\mu = 17°$ approximately). Americans call the half of the angle between the tangents at *F* and *G* the "angle of nip"; it is obvious that the "angle of nip" is equal to the angle *HFK*.

Let the radius of the rolls $= R$, the radius of the sphere to be crushed $= r$, and the distance *ED* between the rolls $= 2d$.

Angle $DAC = 90° - ACL = 90° - KFA = \theta°$, where $\theta = \tan^{-1}\mu$,

$$\cos\theta = \frac{LA}{CA} = \frac{R+d}{R+r}; \ (\cos\theta° = \cos 17° = 0{\cdot}96),$$

$$R + d = (R + r)\cos\theta,$$

$R(1 - \cos\theta =)\ r\cos\theta - d$; or taking the above value for θ

$$0{\cdot}04\,R = 0{\cdot}96\,r - d,$$

$R = 25\,(r - d)$ approximately.

So that the minimum admissible radius of the roll is hereby determined; it is best expressed by saying that the diameter of the roll should be at least 25 times the difference between the mean radius of the pieces to be broken and the mean radius of the particles in the crushed product. In this formula the weight of the sphere, which is relatively quite unimportant, is not taken into account.

Rittinger gives the formula (using the above notation) $R > 18\,(r - d)$, and thus takes a higher coefficient of friction than is given above. His coefficients are $\mu = 0{\cdot}35$ and $\theta° = 19°\,15'$. The exact determination of these coefficients is a matter of considerable difficulty, but it is best to keep on the safe side and to make the rolls sufficiently large. The formula indicates that the rolls must be greater, the greater is the degree of reduction, and it shews that if a very large amount of reduction were required at one operation the rolls would have to be inconveniently large; these machines are therefore only used for medium crushing, beginning with material that has previously been broken in rock-breakers; when this is not done it is better to perform the crushing in two operations, using one pair of rolls for the coarser and another for the finer breaking. In order to enable rolls of smaller diameter to be used, some device for increasing the friction between the rolls and the mineral to be crushed is also at times adopted; this is best done by furnishing the rolls with corrugations or teeth. The first set of rolls or "roughing rolls" are often so treated.

The so-called **Cornish rolls** present the oldest form of this class of machine, and appear to have been first used in Cornwall about the beginning of the 19th century. They are shewn in Figs. 108 to 110[1]. The rolls or rollers themselves consist of two cast iron cylinders, *A*, *B*, keyed or wedged on to the shafts, which may be either round or square. An outer wearing shell of hard steel or good chilled iron is slipped over the roller bodies and secured by means of iron keys or wooden wedges as shewn on an enlarged scale in Fig. 110. One of the shafts is positively driven (in Cornwall often by means of a water wheel with which it is connected by gearing), the other is sometimes driven only by friction, but is now usually driven by gearing, a spur wheel being keyed on each of the roller shafts, this being the arrangement shewn in the figure. These spur wheels must have involute teeth so as to be in gear whatever the distance apart of their axes may be; both the spur wheels and the rollers are best made of equal diameter so as to keep the peripheral speeds of both rolls equal, and thus avoid anything of the nature of a rubbing or shearing action which would produce much dust instead of the granular product that is aimed at. Sometimes the directly driven roll is, as shewn in the plan (Fig. 109), rather wider than the other; this roll is carried in fixed bearings. The bearings of the other roll consist of blocks sliding in guides; they are pressed inwards by means of bell-crank levers, *C*, the longer arms of which are loaded, usually by means of a box filled with stones, scrap iron, etc. The longer arm is usually about 9 times the length of the shorter one, and the weight may vary from ½ ton to about 2 tons according to the material to be crushed. The object of this arrangement is that the rolls may be kept pressed towards each other by a uniform pressure, but that if anything too hard to crush gets into the rolls, it can force these apart and drop through without injuring the machine. The rollers are carried in a substantial frame. Above them is suspended a hopper to receive the material to be broken, and the crushed mineral drops into an inclined trommel, *D*, which screens out all that is sufficiently fine. The oversize drops into a "raff wheel," *E*, a wheel furnished as shewn with internal buckets by which the refuse is elevated and returned by the short *F* to the hopper to go through the rolls again. The following table[2] gives particulars of the dimensions and other details of a number of characteristic Cornish rolls in use in various places about the year 1878:

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[1] *Proc. Inst. Mech. Eng.*, "On the mechanical appliances used for dressing Tin and Copper Ores in Cornwall," by H. T. Ferguson, 1873, p. 119.

[2] *A Descriptive Treatise on Mining Machinery*, S. S. André, Vol. II. 1878, p. 186.

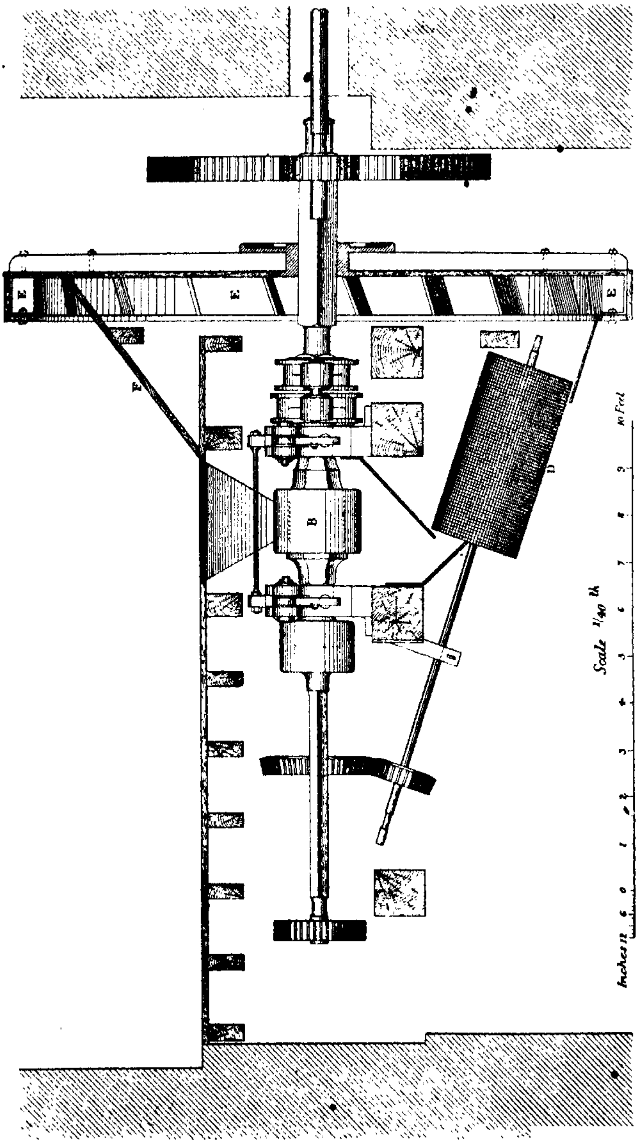

Fig. 108. Cornish rolls. End elevation.

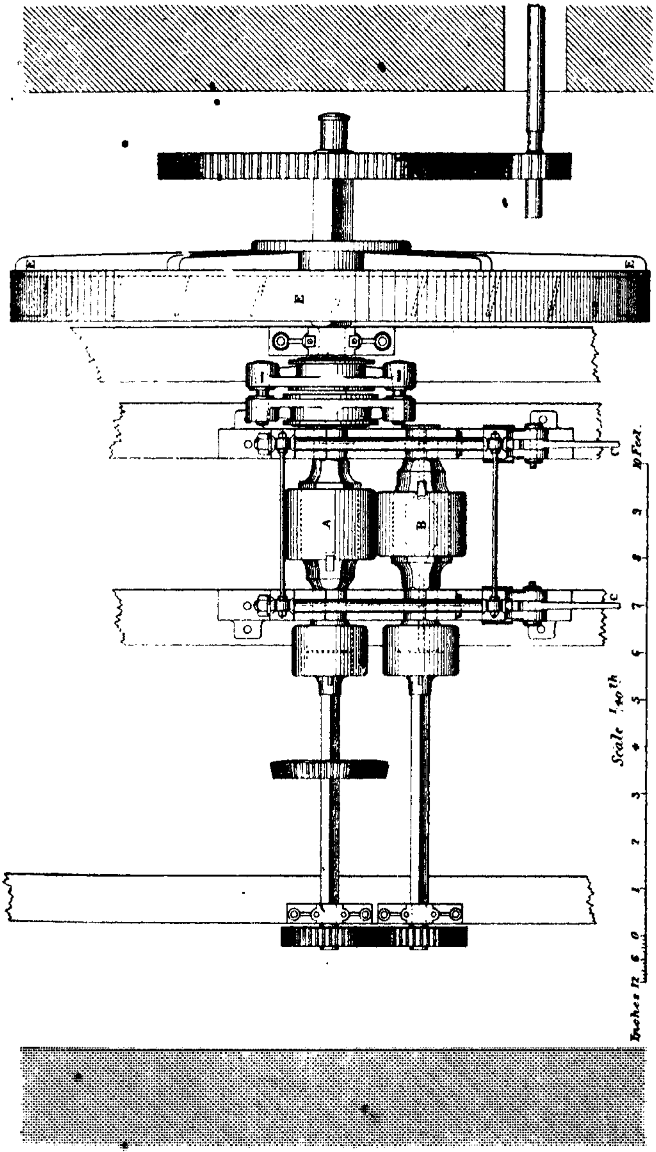

Fig. 109. Cornish rolls. Plan.

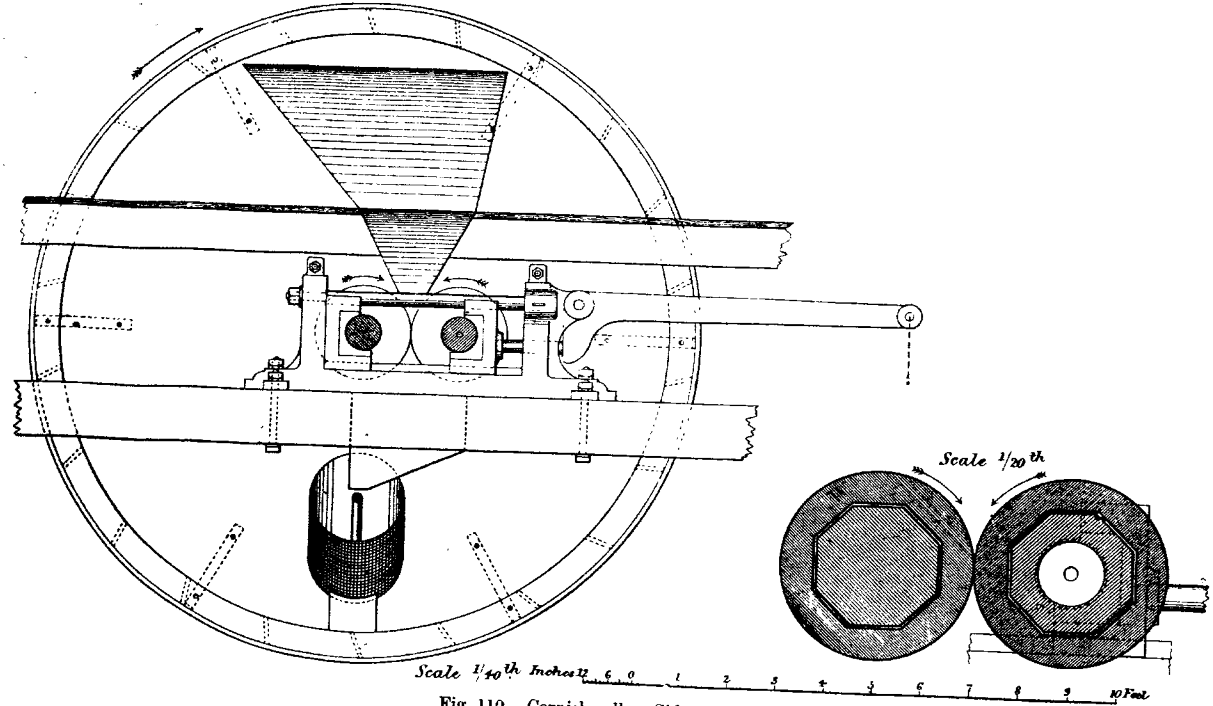

Fig. 110. Cornish rolls. Side elevation.

Name of mine	Rolls				Total pressure on Rolls	Trommel				Raff-wheel diameter	Horse-power required to drive	Tons crushed per 10 hours	Cost of crushing per ton
	Diameter	Length	Revolutions per minute	Crushing area per minute		Diameter	Length	Holes per sq. inch	Revolutions per minute				
	inches	inches		sq. inches	cwts.	inches	inches			feet			pence
Grassington mines......	27	12	5½	5,593	91	21	48	6¼	37	14	—	80	—
Minera	14	14	8	4,920	73⅓	24	42	9	48	10	6	20	2½
Cwmystwith No. 1......	27	14	4	4,748	78	20	33	9	24	16	—	32	2½
Cwmystwith No. 2......	27	14	4½	5,341	85	24	36	9	24	16	—	35	2½
Goginan	30	14	5½	7,254	39	20	39	9	36	16	—	30	2¾
Cwm Erfin	27	14	7½	8,902	293	26	32	9	30	16	—	20	8
Lisburne No. 1	27	15	6	7,632	180	22	36	12¼	30	16	—	42	2½
Lisburne No. 2	27	15	6	7,632	224	22	36	12¼	30	16	—	42	2½
Derwent	27	14	7	8.309	227	22	60	16	—	15	—	60	2¾
Goldscope	14	18	14	11,060	6	—	—	—	—	—	—	25	2½
East Darren	30	18	6	9,996	207	24	36	16	45	16	—	25	2⅝
Cefn Cwm Brwyno ...	20	13	5	4,080	84	20	48	16	27½	14	—	20	2½
Lisburne No. 3	18	16	8	6,432	169	22	36	25	30	16	—	42	2½
Llandudno	18	15	15	12,705	61	—	—	—	—	—	—	30	—
Wheal Friendship	23	12	10	8.670	123	24	36	36	30	13	13	20	11½
Pontgibaud	25	12	12½	12,075	36	22	44	36	60	15	15	17	2½
Devon Great Consols	34	22	7	16,443	458	24	84	64	21	—	—	65	8¾

Modern rolls are more strongly built and more compact, and are run at considerably higher speeds than the old Cornish rolls. One of the chief structural differences consists in replacing the weighted levers by springs; these are either alternate discs of solid indiarubber and sheet iron or else strong volute or helical springs wound round an iron spindle, the outer end of which is threaded and furnished with a nut so that the springs can be tightened up to any required extent and thus be made to exert the necessary pressure. This is a far neater arrangement than

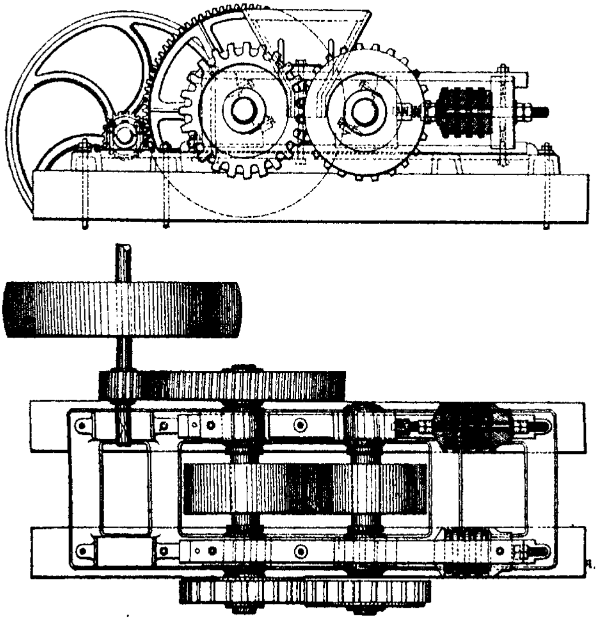

Fig. 111. Geared high-speed rolls. Elevation and Plan.

the old weighted levers, but is mechanically inferior; the weighted levers are practically rigid until the pressure between the rolls exceeds the load on the levers, but when springs are used, these will always yield to some extent in proportion to the pressure between the rolls, so that the newer method involves greater irregularity in crushing than is attained when a heavy weight, hydraulic pressure, or a rigid construction is employed.

All these rolls are intended primarily for medium crushing, that is to say, for yielding a product only moderately fine, crushing say down to about pea size.

More recently however rolls have been extensively used also for fine crushing, down to about 30 mesh; the general design remains about

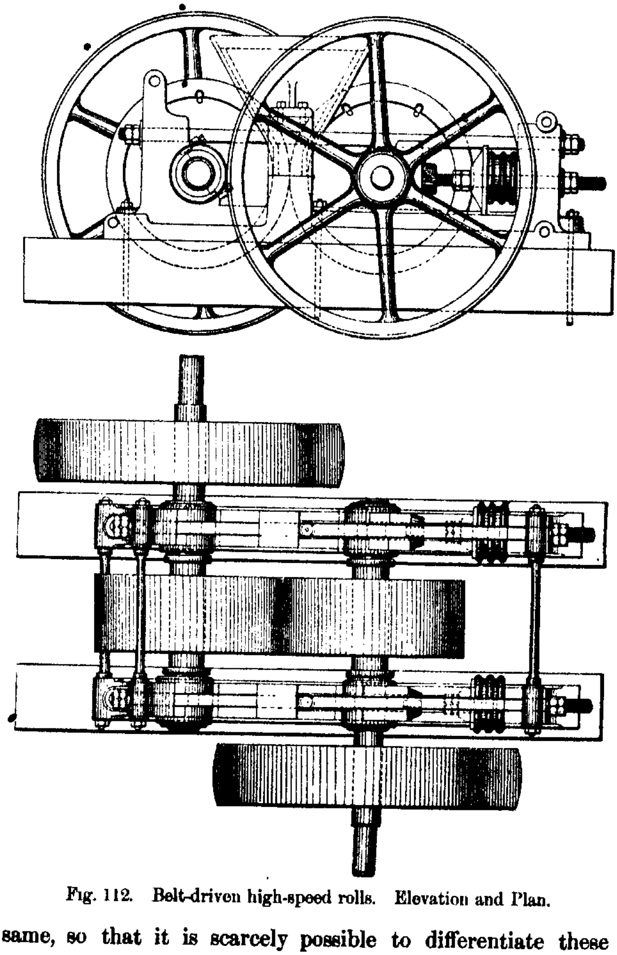

Fig. 112. Belt-driven high-speed rolls. Elevation and Plan.

the same, so that it is scarcely possible to differentiate these two classes. As a rule, however, the peripheral speed of the rolls for fine crushing is much higher. The difference lies largely in more accurate and careful construction and depends to a great extent upon the employment of superior material for the roll surfaces; it is obvious that in order to yield a uniform fine product these latter must be very smooth and true. The rolls must be held together very accurately, either by means of exceedingly powerful springs or else by dispensing with springs altogether. Two sets of rolls as made by Messrs Fraser and Chalmers, Ltd., are shewn in Figs. 111 and 112 respectively. The general construction is very similar in both cases, but the former are geared rolls, one of the roll shafts being driven from a pulley shaft by means of gearing, whilst the other roller shaft is driven by gearing from the first one. In Fig. 112, each roll shaft is driven independently by means of belting, usually by one open and one crossed belt off a pair of pulleys on the same main shaft. The latter construction is well suited to heavy and powerful rolls. In both these rolls solid indiarubber springs are employed.

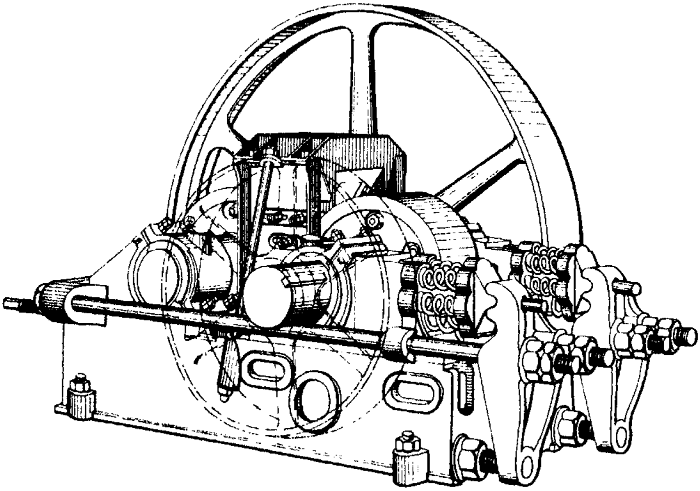

Fig. 113. Belt-driven high-speed rolls. Perspective view.

In Fig. 113 is shewn a set of rolls, belt-driven like the last, but having the sliding bearing pressed towards the fixed bearing by nests of powerful spiral springs, drawn up by a long substantial through bolt.

Krom rolls, manufactured in this country by Messrs Bowes, Scott and Western, Ltd., are used for fine crushing exclusively. These rolls are shewn in side elevation in Fig. 114. The rolls are carried on shafts which are not geared together, but are driven independently by belts, one of which is open and the other crossed. This arrangement enables the two rolls to be driven at different speeds if desired, and this was the original intention. It was found however that when the two rolls are run at markedly different speeds, the action upon the particles of ore is no longer a true crushing one, but shearing is super-induced, thus producing an excessive amount of dust or slimes. It is, on the other hand, an advantage to drive the rolls at very slightly different speeds, say in the ratio of 100 : 101, the difference not being enough to cause shearing in the particles of ore, but merely to ensure that corresponding portions of the roll surfaces shall not always come together, thus producing a more uniform wear of the crushing surfaces. One of the rolls (the right one in Fig. 114) is carried in fixed bearings, the other being supported on swinging arms pivoted on the base plate of the machine, and drawn over by means of batteries of spiral springs which exert a powerful pull through the steel coupling bolts. The method of securing the forged steel roller shells is ingenious. The body consists

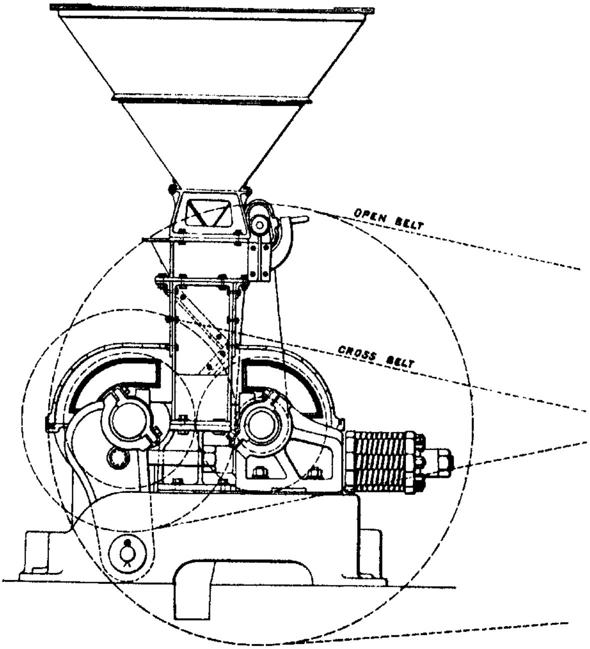

Fig. 114. Krom rolls. Elevation.

of two truncated cones, tapering inwards as shewn in Fig. 115. One of these is forced on to the shaft by hydraulic pressure, and is thus firmly fixed; the other slides on the shaft, being rotated by a feather on the shaft which fits into a groove in the conical boss, the two cones being held together by powerful bolts. The inside of the shell is turned to fit this double-coned body. When it is required to renew a shell, the bolts are loosened, the sliding half of the boss is drawn out, the old shell is slipped off and the new one slipped into place, the sliding cone is replaced and the bolts tightened up. By drawing up these bolts the shell is held in place by friction against the boss. A worn out shell can thus be replaced easily and rapidly, but is quite firmly held in place when the machine is in use. The shells are usually 26 inches in diameter by 15 in length; they are run at from 80 to 100 revolutions per minute, and they require about 10 to 12 H.P. Such a pair of rolls will crush about 2 tons of average ore per hour to about 30 mesh. It is said that a pair of shells will crush about 15,000 tons of average ore before they are worn out. For successful fine crushing it is especially important that the ore to be crushed should be delivered on to the rolls as uniformly as possible, and for this purpose an automatic feeder is mostly employed. The feeder used with the Krom rolls takes the form of a vibrating tray beneath a hopper, into which the ore is charged, usually by hand.

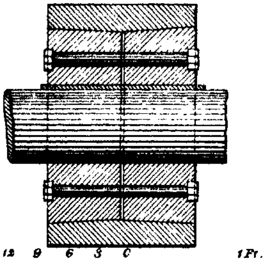

Fig. 115.
Section through a Krom roll.

The Denver Engineering Works Co. make a medium crushing roll in which one of the roller shafts is fixed whilst the other is carried on a swinging lever, somewhat like the Krom roll, except that the springs act in a vertical line. These rolls, shewn in perspective in Fig. 116 and in plan and side elevation in Fig 117, are made in three standard sizes, namely:

Width of face	Diameter of roll	Revolutions per minute
inches	inches	
12	20	100 to 150
14	27	75 to 125
16	36	50 to 75

The makers state that according to a series of experiments by Mr Philip Argall, there is a definite peripheral speed at which rolls do their best work, this speed depending upon the size of the ore to be crushed, as shewn in the following table:

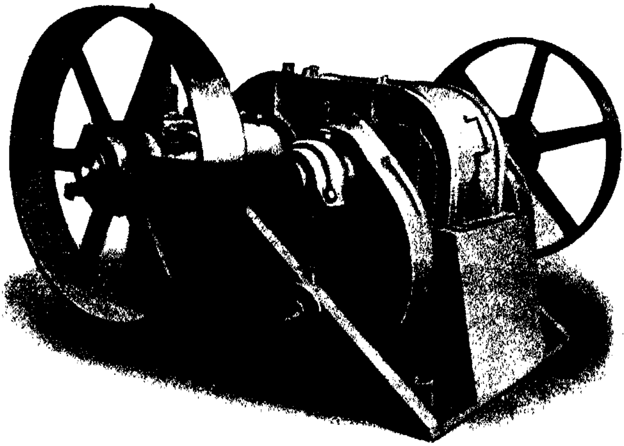

Fig. 116. Medium crushing rolls. Denver Engineering Works. Co. Perspective.

Diameter of pieces of ore to be crushed	Peripheral speed of rolls
inches	feet per minute
1·25	350 to 400
0·25	350 to 600
0·05	700 to 750
0·035	1000

The data are shewn more fully in the graphic diagrams issued by them, Figs. 118 and 119.

Mr Argall[1] has recently designed rolls in which the two bearings of the free roll are situated at either end of a U-shaped yoke, the pressure

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[1] *Trans. Inst. Min. and Met.* Vol. x. 1902, p. 234.

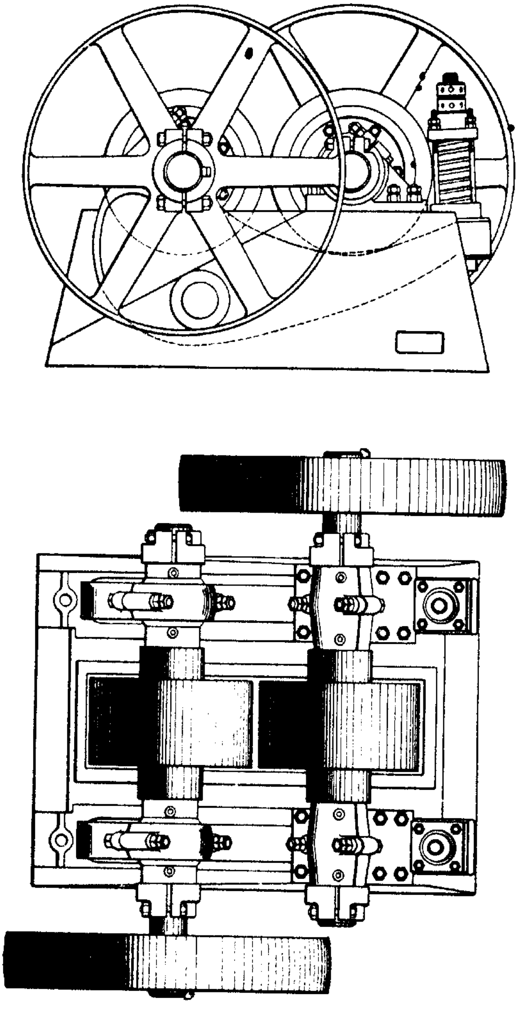

Fig. 117.
Medium crushing rolls. Denver Engineering Works Co. Plan and side elevation.

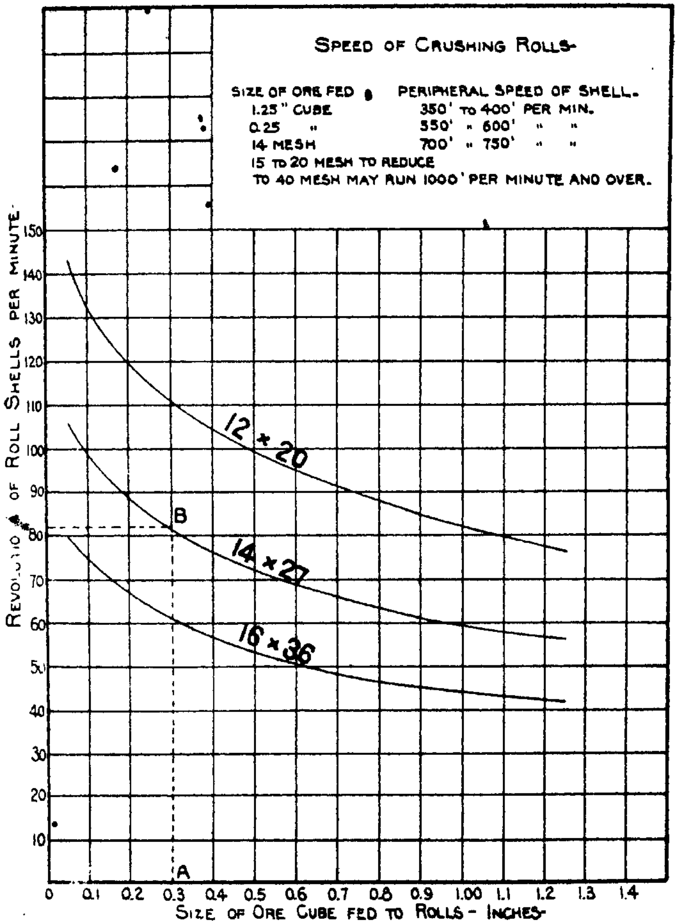

Fig. 118. Diagram of speed of rolls.

being applied through the medium of springs and tie bolts to the central portion of the yoke; by this means he insures that the pressure upon both bearings shall be equal, and at the same time maintains the fixed axis and the free axis always strictly parallel to each other.

A somewhat similar arrangement is adopted by the Humboldt Engineering Works, as shewn in Fig. 120.

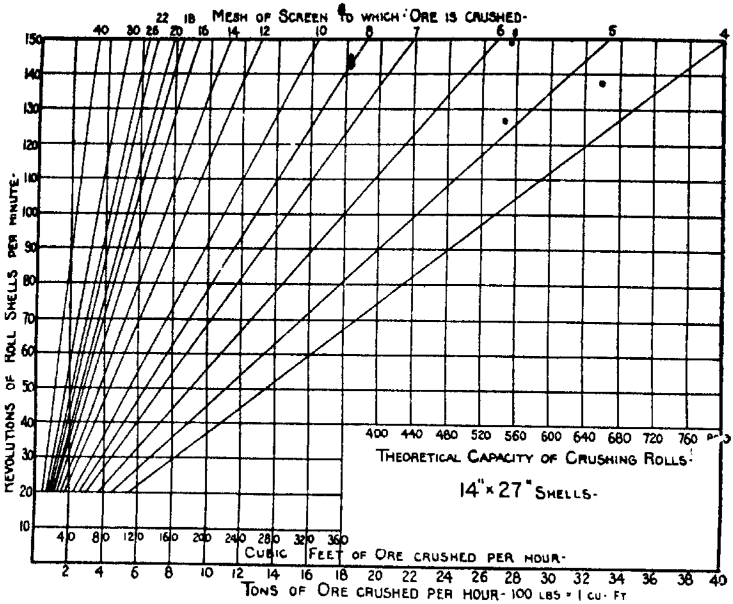

Fig. 119. Diagram of theoretical capacity of rolls.

The following table shews some of the leading sizes according to the last-named makers:

Rolls		Revolutions per minute	Horse-power required	Crushing capacity per hour
Diameter	Length			
inches	inches			cwt.
10¼	10¼	130	8	25
12⅝	10¼	110	4½	43
16	10¼	90	6½	65
21½	10¼	75	9½	102
27½	11	45	12½	145
37½	12	45	16	200

The above production is for rolls set $\frac{3}{16}$ inch apart; the smallest will take $\frac{3}{4}$ inch cubes and the largest $2\frac{1}{4}$ inch. This firm supplies tyre grinders by means of which the surfaces of the tyres can be ground and kept true whilst the rolls are in operation. It will be noticed that the rolls are, as is now often the case, supplied with material by means of an automatic shaking feeder. Another modern set of rolls designed by Edison for fine crushing is shewn in Fig. 121.

In some of the fine crushing rolls made by the Denver Engineering Works Co. both rolls are carried in fixed bearings; one of these is permanently bolted to the base plate of the machine; the other slides in guides and can be fixed by means of bolts at any desired distance from the first-named so as to crush to any desired degree of fineness. The arrangement is shewn in plan and elevation in Fig. 122 and in perspective in Fig. 123. They are made in the same three sizes as the rolls with springs. The makers of these rolls hold that springs are not required for fine crushing rolls, because in the first place any substance likely to injure the rolls will have been removed in the course of the screening which should always follow coarse crushing and precede fine crushing, and in the second place should anything of the kind get in, no harm will be done provided every part of the machine is of ample strength to resist the maximum strain that

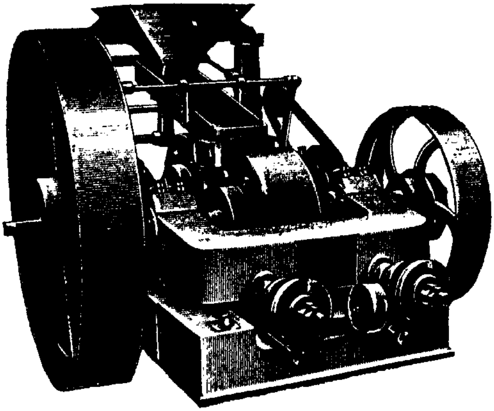

Fig. 120. Fine crushing rolls. Humboldt Engineering Works.

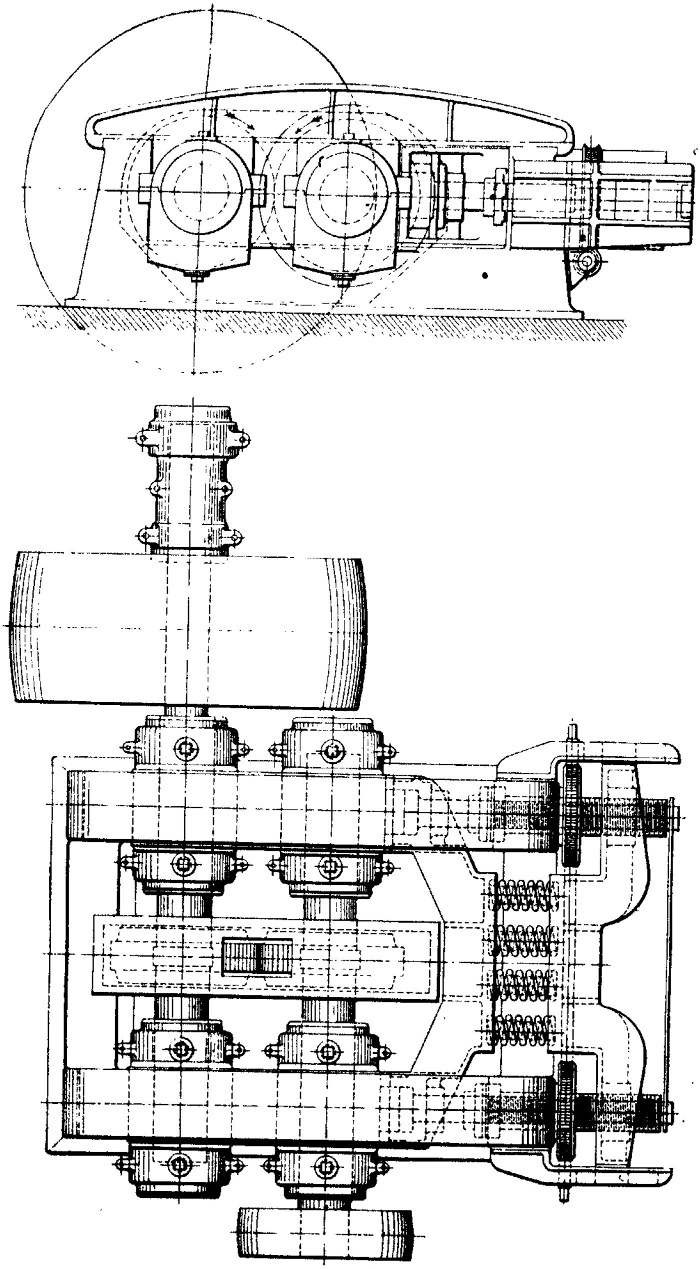

Fig. 121. Modern Edison high-speed rolls. Elevation and plan.

can possibly be put on by the driving belt. Should any substance then find its way into the rolls too hard to be crushed by them, the rolls will simply be pulled up and the belt thrown off. It is claimed, and

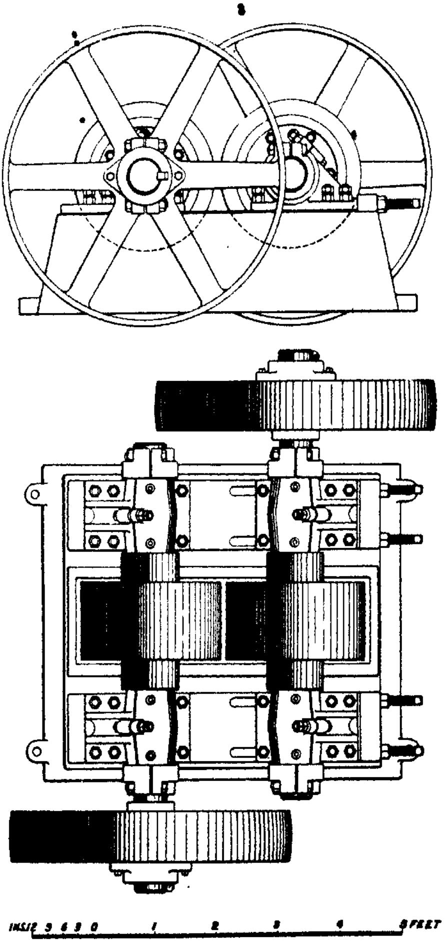

Fig. 122. Rolls with rigid bearings. Plan and elevation.

apparently with truth, that when both rolls are held rigidly the required distance apart, a more uniform product is obtained than when this distance is subject to variation, as it is when springs are employed.

The shells are attached to these rolls in the same way as is used for Krom rolls.

The principle of having both rolls carried in fixed, but adjustable, bearings appears to be sound, though provision must be made so that no serious damage shall be done should an unduly hard piece, e.g. a steel nut, a pick-point, etc., find its way into the rolls. This may be effectively secured by having the driving pulley loose on the shaft, and connected to a driving collar keyed to the shaft by one or two breaking pins, or bolts, as is done in some of the rigid rolls of the Denver Engineering Works Co. This device is illustrated in Fig. 124; the driving pulley there shewn runs loose on the shaft, whilst a collar is keyed on, to which the pulley is secured by bolts of such strength as to shear before any injury is done to the rolls.

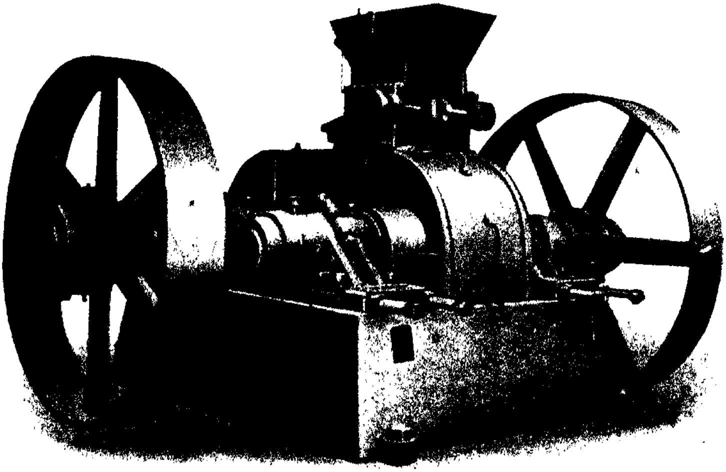

Fig. 123. Rolls with rigid bearings (enclosed). Perspective.

Another method sometimes used is that of having one of the bearings movable, and replacing the springs by a breaking piece, usually a hollow casting arranged so as to break whenever the strain on the rolls exceeds what may be looked upon as the safe limit. By this means the rolls are held rigidly at the desired distance apart, whilst the machine is yet protected from any injurious strain; this is probably preferable to the use of either springs or weighted levers when a uniformly sized product is required.

In modern practice it is usual in large plants to arrange sets of 3 to 5 rolls in series, so as to gradually reduce the mineral to be crushed, all the mineral that is sufficiently fine being screened out between each set of rolls. Such a system of rolls may either be actually set one above the other or the rolls may all be on the ground level, the material being elevated between each pair of rolls. Both modes of construction have their advocates.

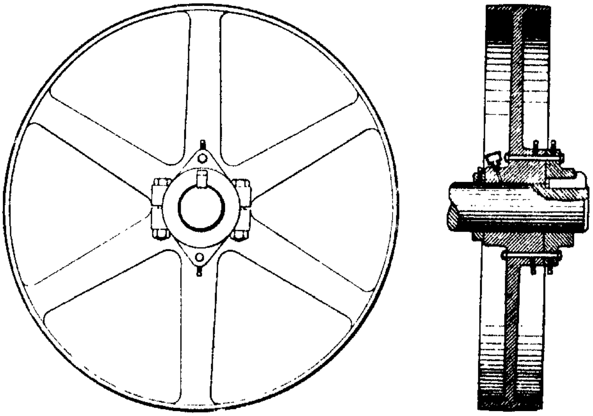

Fig. 124. Safety driving pulley with shearing bolts. Elevation and section.

A typical plant of this kind at the Mount Morgan Mine consists of eight sets of rolls arranged in two series of four each, as follows:

I. 26 in. diameter, 15 in. face, set $\frac{3}{8}$ in. apart, running at 112 revs. per min.
II. 26 „ „ 15 „ „ $\frac{3}{16}$ „ „ „ „ „
III. 30 „ „ 16 „ „ $\frac{1}{16}$ „ „ „ „ „
IV. 30 „ „ 16 „ „ in contact.

The total power consumption (including breakers, etc.) was 100 H.P., and the crushing capacity 125 tons per 24 hours crushed to 20 mesh (0·025 inch). The wear on the roll tyres is given as 0·108 lb. of steel and the cost of the material for renewals as 10·8*d.* per ton crushed.

Mr Argall[1] estimates for a plant to crush 200 tons of ordinary ore to 0·02 inch as follows:

A 12 inch × 20 inch Blake crusher breaking to 1·7 inch cube.

I.	Roll 36 in. diameter,	16 in. face,	35 revs. per min.,	crushing to 0·75 inch.
II.	,, 26 ,, ,,	15 ,, ,,	65 ,,	,, 0·25 ,,
III.	,, 26 ,, ,,	15 ,, ,,	90 ,,	,, 0·1085 ,,
IV and V.	,, 26 ,, ,,	15 ,, ,,	110 ,,	,, 0·02 ,,

He considers that this plant would require 105 I.H.P. The average wear of the shells of such rolls is given as 0·226 lb. of steel per ton of ore crushed.

It is frequently desirable that a machine intended for crushing to a given size shall make as little fine dust as possible; the size limit of the latter is usually taken as between 0·002 and 0·003 inch. Rolls are particularly satisfactory in this respect, delivering a product that is granular rather than dusty.

Thus in crushing down to 0·02 inch with rolls the products should be about as follows on average ores:

Between 0·02 in. and 0·01 in.	15 per cent.
Between 0·01 in. and 0·005 in.	20 ,,
Between 0·005 in. and 0·0025 in.	35 ,,
Below 0·0025 in.	30 ,,

It has already been pointed out that when an exceptional degree of reduction is required with but few passes, rolls provided with corrugations or projections must be employed so as to produce a positive grip on the material to be crushed instead of relying on friction alone. This is especially the case when preliminary coarse crushing has to be performed, and is more suitable for material that is comparatively easily broken. A typical example is shewn in Fig. 125 which represents a pair of tandem crushing rolls, the upper fitted with teeth and the lower corrugated. These were built by Messrs Head, Wrightson and Co., Ltd. for breaking coal, and will reduce pieces up to 2 feet cube down to 2 inches cube at the rate of 50 tons per hour.

These teeth are made of various forms, and are often built up of a series of rings like toothed wheels threaded upon the body of the roll. Corrugated rolls are often used in the first crushing of ordinary veinstuff, such rolls being often spoken of as "roughing rolls."

Edison has used corrugated rolls for fine crushing, the teeth of his rolls being saw-shaped in cross-section. Such rolls are shewn in Fig. 121; they have been made from 24 inches diameter by 6 inches face up to 36 inches diameter by 8 inches face, and have been run at about 200 revolutions per minute. The rolls are set so that the points of the corrugations are $\frac{1}{16}$ inch to $\frac{3}{16}$ inch apart, but they are forced further apart by the stream of ore passing between them. This is the system of fine crushing

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[1] *Trans. Inst. Min. Met.* Vol. x. 1902, p. 234.

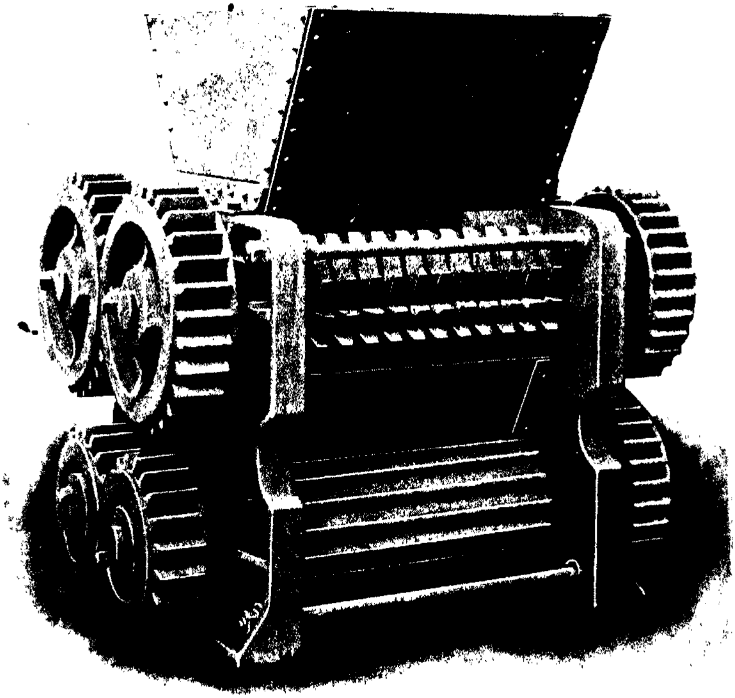

Fig. 125. Toothed rolls for breaking coal.

usually spoken of as choke-feeding, where a body of material is forced between the rolls considerably in excess of what they can crush at once to the required degree of fineness, that which is sufficiently fine being screened out, and the oversize returned to the rolls.

A set of Edison rolls 32 inch × 8 inch using 32 I.H.P. will reduce about 30 tons of cement clinker per hour to a fineness of 87 per cent.

through a 200 mesh screen; while a set 36 inch × 8 inch will reduce from 80 to 100 tons of ironstone per hour to 30 mesh fineness.

The earlier and smaller Edison rolls for fine crushing were drawn together by an ingenious multiplying gear of steel wire ropes[1] actuated by a pneumatic cylinder, but the wear on the ropes was too great, and in the larger modern rolls this rope system is replaced by powerful springs as shewn in Fig. 121. These springs are so arranged that the rolls may be set up to compensate for wear on the corrugated plates without the spring pressure being disturbed, or the spring pressure may be adjusted at will without the set of the rolls being disturbed.

In this connection a decidedly aberrant type of rolls, which work on an entirely different principle to ordinary rolls, may be considered here, this being the **Edison Giant Rolls**. A set, erected in New Jersey, were 5 feet in diameter by 5 feet face, studded with projecting teeth 2 inches high and with several rows of "slugger" knobs 4 inches high. The rolls are faced with chilled cast iron plates, put on in segments and bolted to the iron cores with bolts having countersunk heads. The rolls are set in massive fixed bearings with centres 7 feet 2 inches apart, so that the faces of the rolls are 14 inches apart, leaving a clear space of 6 inches between the points of the knobs. The rolls run at 150 revolutions per minute, the total weight of the moving mass being about 75 tons. There are driving pulleys on each of the roll shafts, the latter being 15 inches in diameter. These pulleys are not however keyed to the shafts but are held on by means of powerful friction grips consisting of band brakes tightened up by strong springs. The rolls are driven by an engine of 80 H.P.; when running empty only about 50 H.P. is consumed. The rock to be broken is fed in by skiploads of 6 to 7 tons at a time, some of the pieces weighing up to 5 tons. When this load of rock is dropped upon the rolls, the speed of the rolls is somewhat checked so that the pulleys slip upon the shafts. In doing so the vast momentum stored up in the moving masses is utilised and under this force the teeth and sluggers impinge upon the rock which is thus rapidly broken down to about 10 inches cube. As soon as the load of rock has passed through, the pulleys grip upon the shafts again, the rolls are accelerated up to their full speed and are then ready for another load. The capacity of these rolls is 300 tons per hour. The cost for wear of plates is said to be 0·1*d.* and for other repairs 0·14*d.* per ton of ore crushed.

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[1] *Trans. Inst. Min. Met.*, "Dry Crushing of Ores by the Edison Process," by W. Simpkin and J. B. Ballantine, Vol. XIV. 1904, p. 70.

It will be seen that the breaking is done wholly by impact, and that the momentum stored up in the moving rolls is utilised to obtain the necessary power.

The material broken in these rolls is further reduced in rolls furnished with knobs, followed by others with deep corrugations; these do not differ from ordinary rolls except in their great size and high speed of running.

Even larger Giant rolls, 7 feet diameter by 7 feet face, in which the weight of the moving masses exceeds 100 tons, as shewn in Fig. 126, have been erected at Dunderland to deal with blocks of ore up to 8 tons in weight; instead of the friction grips above described each roll is worked by an independent engine, which slows up when a large piece of rock is being crushed. The points of the knobs in these rolls are 8 inches apart; they absorb about 130 H.P. and have in actual practice crushed over 250 tons per hour, and could undoubtedly do even more.

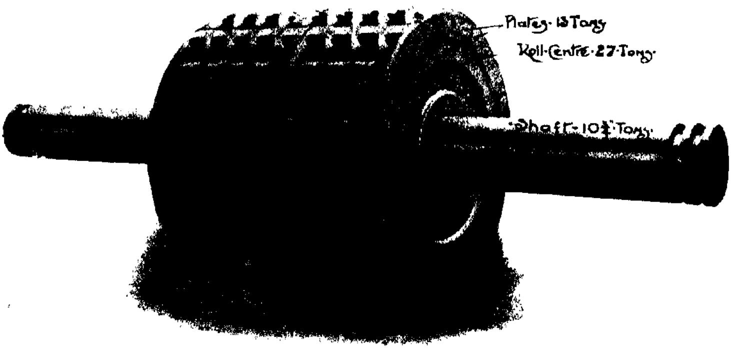

Fig. 126. One of a pair of Edison "Giant" rolls.

The method in which these rolls are arranged and combined with others for the successive reduction of such very large blocks, down to about ¾ inch cube is illustrated in Fig. 127, which represents a somewhat smaller plant used for crushing limestone for the manufacture of cement. **B** is the hopper made of heavy castings into which are tipped the blocks of stone, which may be up to 5 tons in weight; the giant rolls **C** are 5 feet in diameter by 5 feet face, of the above described construction; the stone is broken by these rolls to about 8 inches

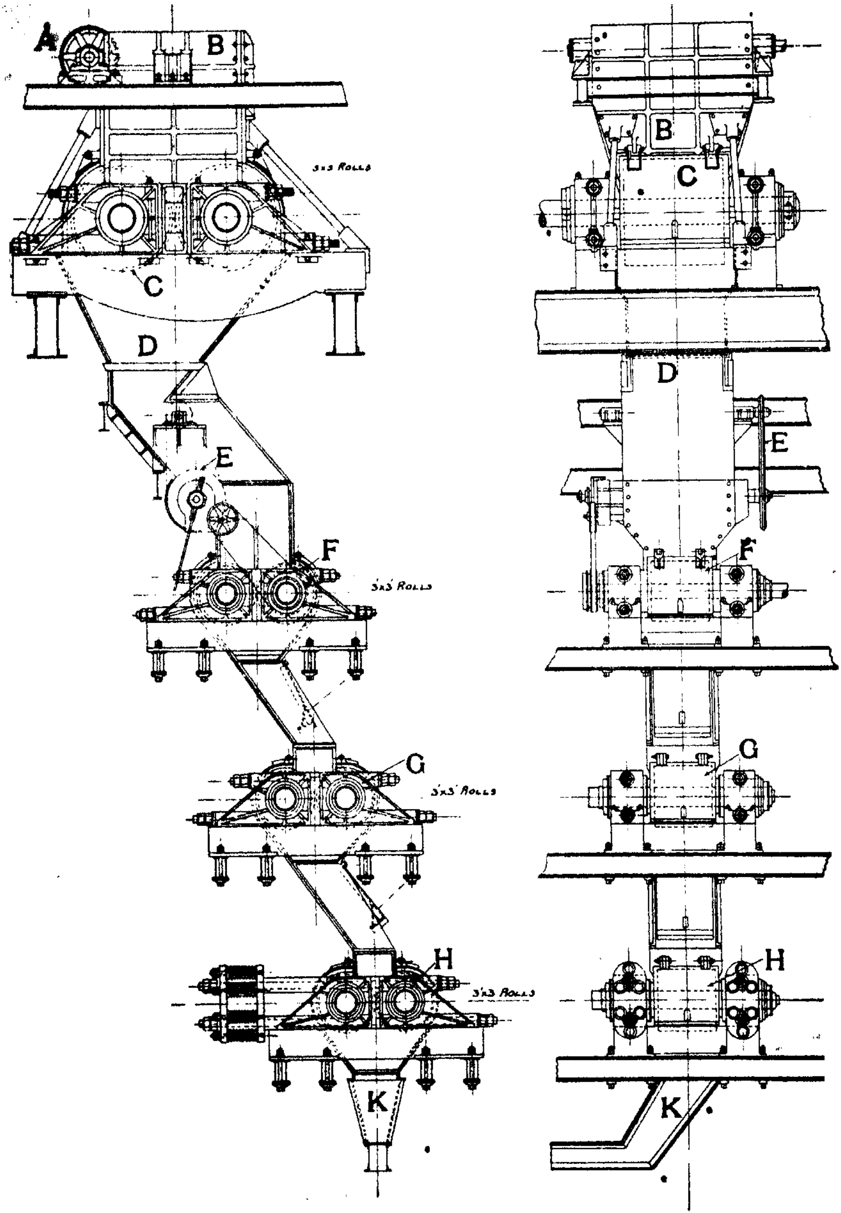

Fig. 127. Arrangement of an Edison coarse-crushing plant.

cube, and then drops into the hopper **D**, from which the feeder **E** delivers it uniformly to rolls **F**, whence it passes in turn to rolls **G** and **H**; these three rolls are all similar, 3 feet in diameter by 3 feet face, with longitudinal grooves or corrugations. The two upper rolls run in fixed bearings, each roll shaft being driven by gearing, a breaking piece being inserted for safety; the lowest roll **H** is furnished with springs. All these rolls run at about 130 revolutions per minute. The total capacity of the plant is 250 tons of limestone per hour broken down to ¾ inch or rather smaller.

C. FINE CRUSHING.

One of the most widely used of fine crushing machines, namely, fine crushing rolls, has just been described, because, as already stated, whilst rolls in their typical form are used for medium crushing, they can also be well adapted for fine crushing, though not so well perhaps for the very finest work. In the same way, stamp mills are essentially fine crushing machines, but one form is used only for medium crushing; its description has, however, been deferred, and will be included under that of stamp mills in general.

Fine crushing is performed either by percussion or by attrition; pressure plays as a rule a subordinate part, though it no doubt comes into action in some grinding machines such as the Chilian mill. In some machines percussive action takes place not against a fixed anvil, but by the collision of particles of the mineral themselves dashed against each other at a high velocity.

The stamp mill. This machine consists essentially of a mechanically worked pestle, the stamp, which is caused to fall upon and pound the mineral placed beneath it. Stamps may be divided into two classes: Gravitation stamps in which the stamp proper is lifted by mechanical means and allowed to drop under the action of gravity, and Power stamps in which the descent of the stamp is due partially or entirely to some mechanical force applied to it.

Amongst the cruder forms of stamping machines may be named several that act upon the principle of the tilt hammer or helve. A very primitive machine of this kind, in use among the Chinese, has been described by the author[1]. In the Lake Champlain district iron ore has been crushed by a battery of tilt hammers falling upon an anvil consisting of cast iron plates with ¼ inch perforations. The tilt hammer type of stamp is now practically obsolete.

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. xx. 1891, p. 324.

The ordinary gravitation stamp consists of a vertical stem terminating in a heavy head; the stem is lifted by suitable machinery (usually by a cam), and the entire stamp is allowed to fall, the head crushing the ore placed beneath it. The simple gravitation stamp is a very old machine; it seems to have been well known in the mining districts of central Germany at the beginning of the sixteenth century and is probably much older. This same type of stamp was introduced into Cornwall probably early in the seventeenth century, and has continued in use in both Germany and England with practically no modification in design, although stone and wood were gradually replaced by iron. Soon after 1850 the stamp was introduced into California, and was there entirely remodelled. The Californian stamp, as it is accordingly called, is the most modern modification of the stamp mill, but its principle is still that of the simple mechanically lifted pestle.

The Saxon and Cornish stamps are characterised by having stamp heads and stamp stems rectangular in cross-section, the stamp being thus unable to rotate, whilst the Californian stamp is circular in cross-section and is capable of rotation about its vertical axis.

The **Cornish stamps**, used chiefly for tin and copper ores, are shewn in front elevation in Fig. 128 and in transverse section in Fig. 129. There are usually four stamps in one battery box or cofer. These consist of wrought iron shanks (or lifters) about 11 to 14 feet in length and about 2 inches × 5 inches in section. These are cast into the rectangular heads, or else keyed in, the former being the more usual plan. The heads are made of white or mottled cast iron, usually about 24 inches in height and from 6 inches to 8 inches × 10 inches to 12 inches in plan, having thus a crushing face equal to 60—100 square inches. To the shank is attached the tappet or tongue, which is capable of sliding up and down the stem and of being fixed by means of a key at any desired point. The stem passes between two pairs of wooden guides, which are merely rectangular beams connecting the battery uprights. The cam barrel which causes the lift of the stamps runs the full length of the entire mill in front of the stamps, and is carried in bearings supported by brick piers or wooden framework. The barrel used to be made of wood, but is now generally a hollow cylindrical casting into which the cams or wipers are fastened by means of keys or wedges; usually there is one cam shaft to every four sets or sixteen heads of stamps. There are usually five or sometimes

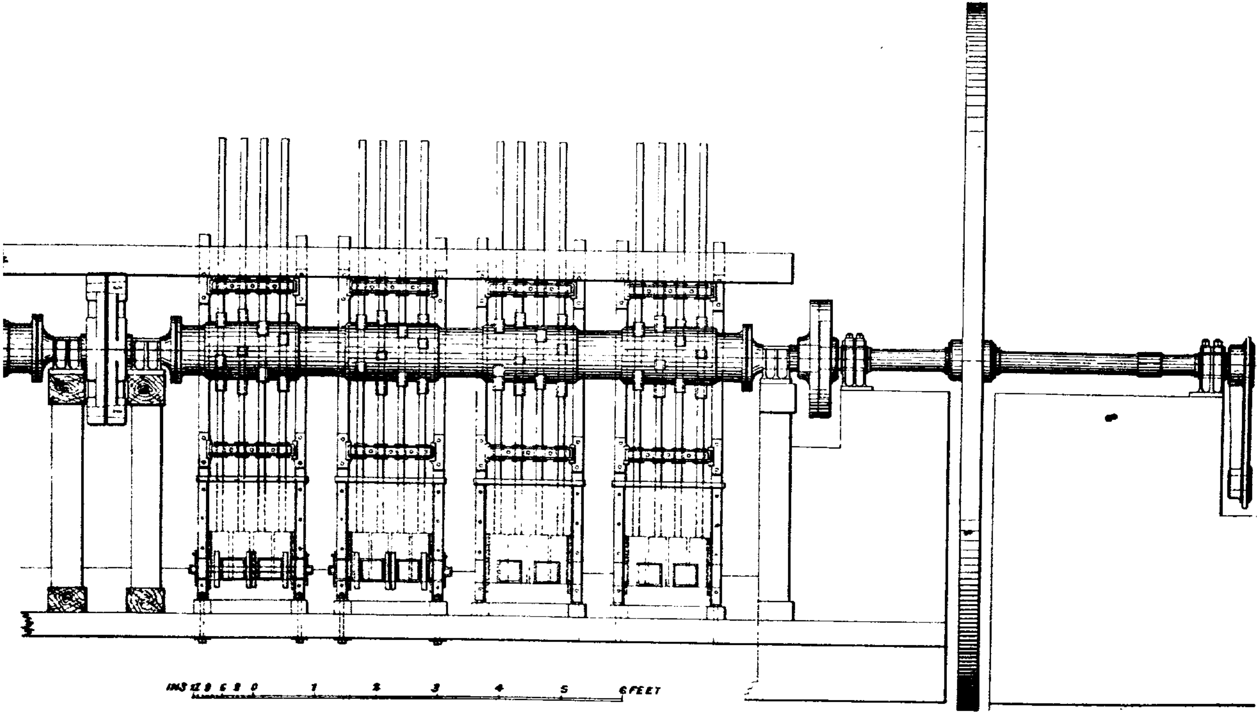

Fig. 128. Cornish stamps. Front elevation.

four cams to each stamp, so that the latter is lifted and dropped five (or four) times for each revolution of the cam barrel. The shape of the cam is that of an involute of a circle, the tangent to an involute being always at right angles to the tangent to its pitch circle. If the pitch circle is therefore of such a size that the vertical tangent to it falls within the vertical travel of the tongue, the surface of the involute in contact with

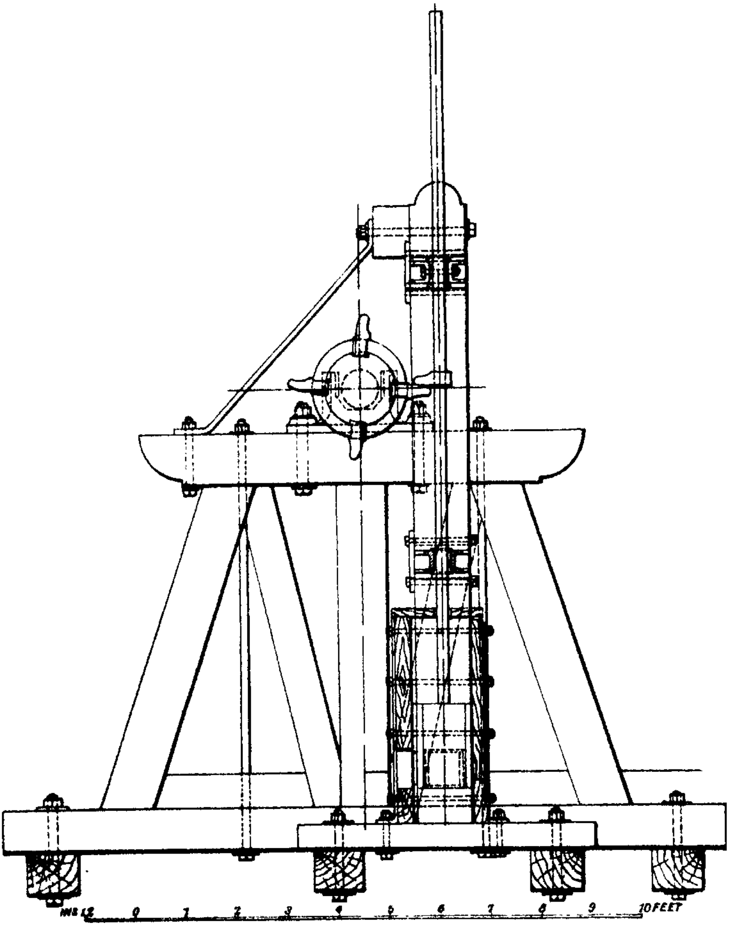

Fig. 129. Cornish stamps. Transverse section.

the latter will always be parallel to it and there will be no side thrust developed in the stamp, the rotary motion of the cam barrel being thus wholly converted into the vertical motion of the stamp. The geometrical construction is shewn in Fig. 130. In this figure, let *O* be the centre of the cam barrel, and *o* the point of contact between cam and tappet. Describe the circle with radius *Oo*, and set off an arc *O*10 equal in length to the desired height of lift of the stamp. Divide this arc into any

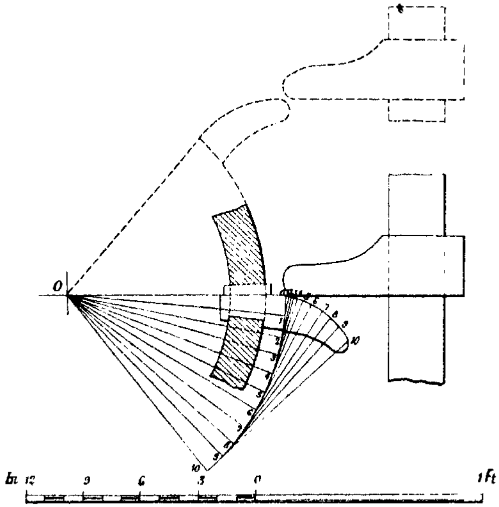

Fig. 130. Development of involute for cam of Cornish stamps.

number of equal parts by the radii *O*1, *O*2, *O*3, ... *O*10; at the end of each radius draw a tangent equal to the lift of the tappet at that point, i.e. equal to the full lift at *O*10 and diminishing by equal amounts until the tangent at *Oo* = 0. Join the points thus obtained, and the resulting curve is the involute which forms the upper surface of the cam. The dotted lines shew the position of the cam and tappet at the top of the lift. In Cornwall the cam barrel is either driven directly by means of a water wheel or else by a steam engine. The speed of rotation is slow, usually 10 to 16 revolutions per minute, the height of lift being about 10 inches. It is usual to cause the corresponding stamps in adjoining batteries to fall in regular succession, the cams being so spaced on the barrel as to produce a uniform torque. Thus, if there are *n* stamps worked off one cam barrel, and if there are five cams to each stamp, the angular distance between successive cams must be $\frac{360°}{5n}$.

The cofer is a wooden box long enough to take the set of four stamps; it has a slot at the back through which the mineral to be crushed is introduced, and a rectangular opening in front in which is fastened the "grate" consisting of a couple of sheets of perforated copper; usually there are also two similar openings, one at either end, also each provided with a grate. There is an inclined plane, known as the "pass," leading from the hopper, or "half-pass," to the slot in the back of the cofer, upon which the mineral is delivered and down which it slides gradually into the stamper box, aided by a gentle stream of water directed upon it, and also by the vibration caused by the action of the stamps. The mineral is crushed either upon a bed plate of cast iron some 3 or 4 inches in thickness, which completely fills the bottom of the cofer, or else the stamps are allowed to "beat their own bed," that is to say, the mineral first introduced is pounded down into the bottom of the mortar by the action of the stamps until a perfectly solid mass of concrete-like stone results, upon which crushing then takes place. The complete weight of each stamp is usually 6 to 7 cwt., it absorbs about 1 I.H.P. and crushes about 10 to 15 cwt. of hard stone in 24 hours. At the Levant mine the heads are 12 ins. by 7 ins. face and 24 ins. high, weighing 4 cwt. The cam barrel makes 12 revolutions and the stamps 60 drops per minute. A battery of 48 heads crushes 1200 tons of hard tin capels in 4 weeks of 6 days each, equal to a little more than 1 ton per head per 24 hours. One head crushes 125 tons of capels in 5 months, at the end of which time it is worn out and returned to the foundry, the average amount of iron worn off being 2½ cwt., costing 6*s*. per cwt. The wear per ton of stuff stamped is thus equal to 2·24 lbs. of cast iron and the cost to 1·44*d*.

Cornish stamps have a low efficiency and require a good deal of attendance; they are, however, relatively cheap to erect, and need but few repairs.

An iron-framed stamp mill of modern German type[1] at the Diepenlinchen mines near Stolberg is shewn in vertical section and plan in Figs. 131 and 132. The foundation is of masonry carrying a cast iron plate upon which the wooden mortar-box is built up; the stamp stems are of wood and the shoes of hard cast iron, their total weight being

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[1] *Ann. des Mines*, "Sur la préparation mécanique des minerais de plomb, etc." by A. Henry, Series VI. Vol. XIX. p. 294, Paris, 1871.

3 cwt. The tappet is a triangular casting the height of which can be adjusted by sliding on a rack and upon which it can be secured at

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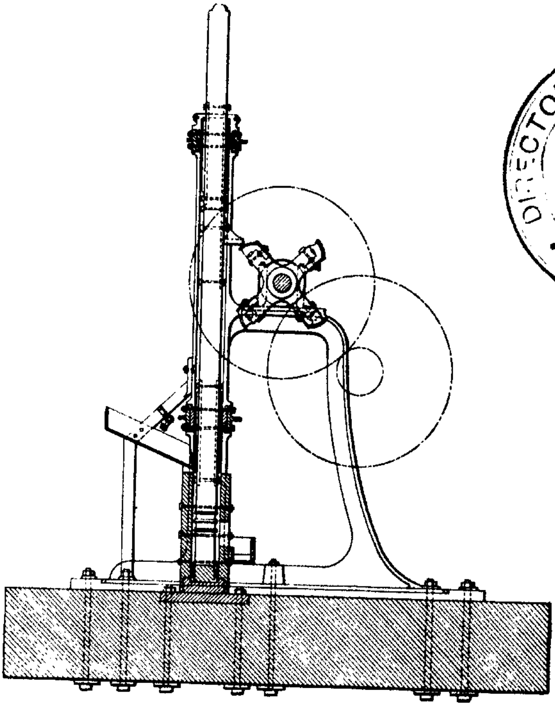

Fig. 131. Modern German stamp mill. Vertical section.

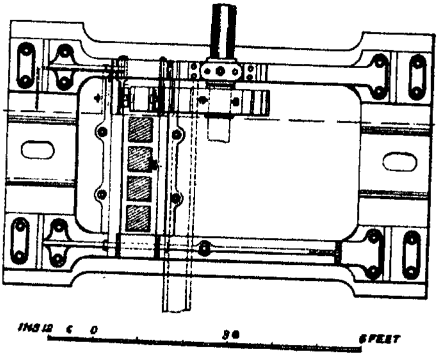

Fig. 132. Modern German stamp mill. Plan.

any desired point. The stamps are lifted by four-armed cast iron cams, the upper wearing faces of which consist of renewable castings made of white iron.

In Germany the simple screen or grate discharge is sometimes replaced by certain other forms. In the so-called "stay-discharge," there is a box outside the grates or screens, from the bottom of which the pulp escapes through nozzles or holes of a size that can be varied as desired, as shewn in Fig. 133. This arrangement has the advantage of delivering a thicker pulp and thus of decreasing the consumption of water; it is said that only half as much water is required per ton of ore as when the simple grates are used, for the same degree of comminution.

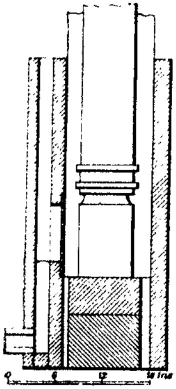

Fig. 133. "Stay" discharge of stamp mill cofer. Vertical section.

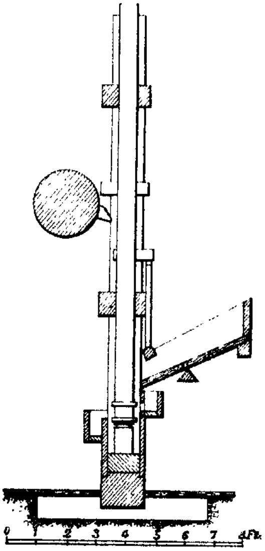

Fig. 134. "Flush" discharge of stamp mill cofer. Vertical section.

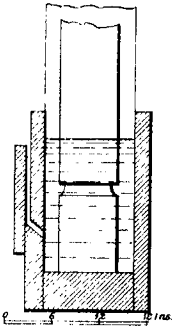

Fig. 135. Improved "Flush" discharge. Vertical section.

Another system, the "flush-discharge," does away with screens altogether, the fineness of crushing being regulated by the height to which the pulverised mineral has to be lifted before it can escape, and the rate at which water is allowed to flow through the battery box. An old form is shewn in Fig. 134, which shews an old German battery box with open-topped cofer over the edge of which the discharge takes place. With a flow of water of between 0·4 and 0·8 cubic foot of water per minute.

a depth of discharge of between 15 and 18 inches corresponds to a diameter of about 0·04 inch in the escaping particles, whilst a depth of 8 inches corresponds to about 0·2 inch diameter. An improvement

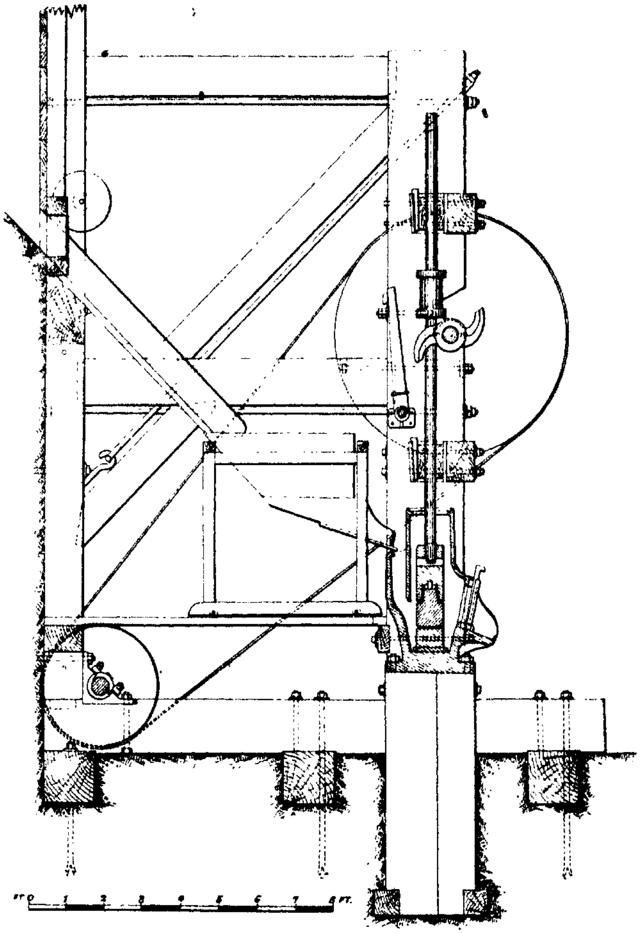

Fig. 136. Californian stamp mill. Vertical section.

is represented in Fig. 135, in which the discharge takes place through a channel about ½ inch to ¾ inch wide, running the full length of the cofer; the opening of this channel is about 4 inches above the bottom of the box, and its depth varies from 9 inches to 15 inches. The fineness of the material discharged depends both on the depth of the channel and upon the flow of water, the latter averaging 0·25 cubic foot per minute in ordinary cases.

The **Californian stamp** differs from the Cornish stamp mainly in that the head, stem and tappet are circular, and are lifted by cams which are set on one side of the axis, thus causing the entire stamp to rotate slowly about that axis so as to render the wear of the stamp more uniform; the surface upon which crushing takes place no longer consists of one single die, but there is a separate die under each stamp; the mortar box is a massive iron casting, and there are usually five stamps

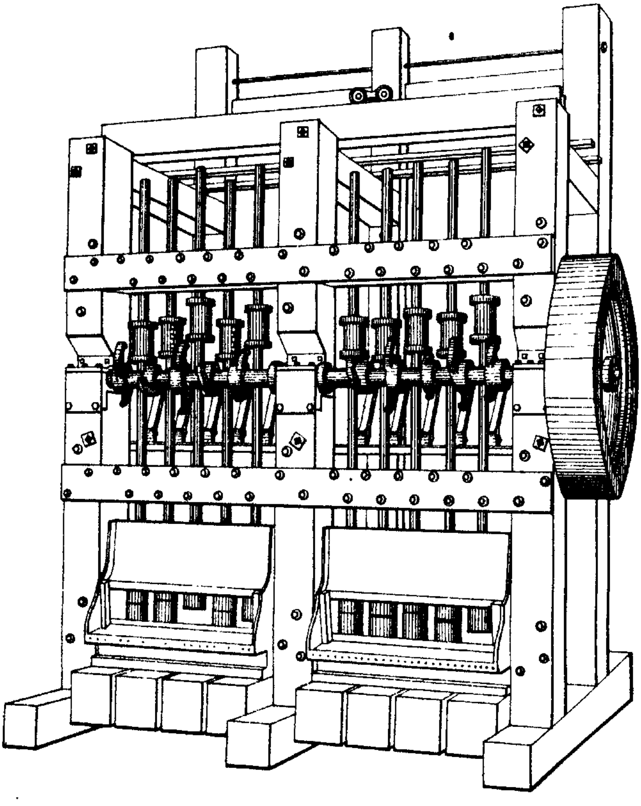

Fig. 137. Californian stamp mill. Perspective.

to each mortar. Californian stamp mills are chiefly used for the crushing—and generally the simultaneous amalgamation—of gold ores, and the entire operation thus becomes practically a metallurgical process, the consideration of which would be out of place here[1]. This stamp will be treated here simply as an appliance for fine crushing. A general section of such a mill is shewn in Fig. 136, and a perspective view of the same in Fig. 137, this being one designed by the author for use in South Africa. A ten stamp battery, differing from the last only in some minor details, but shewing the ore hoppers behind it, built by Messrs Edward Chester and Co., Ltd., is shewn in Fig. 138. In this latter each head of 5 stamps is driven

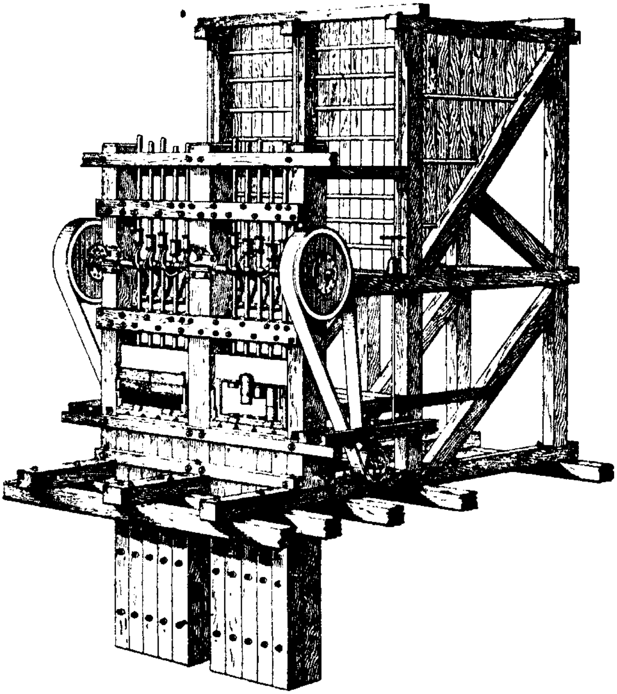

Fig. 138. Californian stamp mill. Perspective.

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[1] For an exhaustive description of the Californian stamp mill, as applied to the treatment of gold quartz, the reader should consult the author's *Handbook of Gold Milling* (Macmillan & Co.).

independently, whilst in Fig. 137 each ten heads is so driven. The mortar is supported upon a mortar block, which is mostly formed of massive wooden baulks set on end and firmly bolted together; the more modern type of block, built of good bricks laid in cement or moulded in concrete, with about 6 inches of wood on top, or sometimes with only a sheet of indiarubber on the concrete, is, however, far preferable. An iron-framed mill with such concrete mortar blocks is shewn

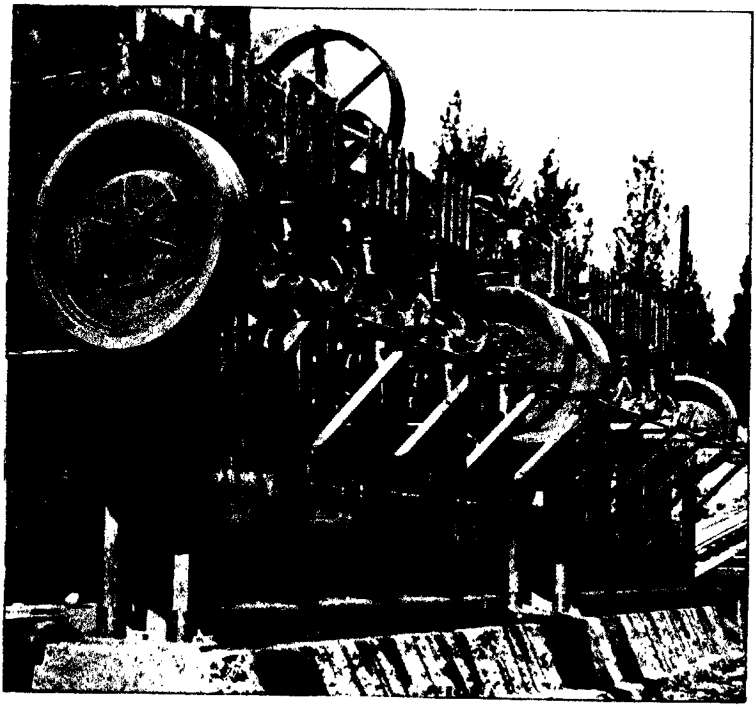


Fig. 139. Iron-framed Californian stamp mill. Perspective.

in Fig. 139, which represents a 40 stamp mill being erected at the Pole Star Mine, Grass Valley, Nevada Co., California. In any case the mortar block must rest upon a thoroughly sound foundation. The mortar consists of a single casting having an opening in front into which fits a screen frame carrying the screen, which latter is mostly now-a-days made of woven steel wire; slotted or more rarely punched steel plates are also used. At the back of the mortar is a feed shoot, through which the ore to be crushed is delivered upon the dies. To the front of the mortar a renewable shoot or apron is bolted, which receives the pulp as it passes through the screens. The most important dimensions of the mortar box are its width at the level of the lower edge of the screens, and the depth of the top of the dies below this same level, this depth being usually spoken of as the "depth of discharge." This necessarily increases as the dies are worn down, and various devices are used to keep it approximately constant: this may be done by raising the dies by the insertion of a false bottom. Another method consists in introducing a so-called "chuck block" below the screen frame, which can be removed and a thinner block substituted as the dies wear down. A mortar with such a chuck block, that is very much used in the United States of America and in the Transvaal is shewn in section in Fig. 140, whilst Fig. 141 represents another modern form by Messrs Bowes, Scott and Western, Ltd., in which the constructional details are shewn. The lower the depth of discharge and the narrower the mortar, the greater will be the crushing capacity of the stamp mill, other things being equal. When a stamp mill is used for amalgamating as well as for crushing, neither

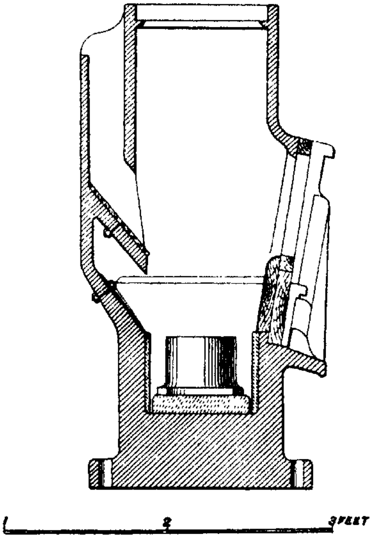


Fig. 140. Mortar. Vertical section.

width nor depth of discharge may be allowed to fall below certain dimensions, dependent upon the character of the ore treated and the system of amalgamation adopted. When the mill is used for crushing alone, both of these dimensions should be kept as low as possible, the chief point to be borne in mind being that when they are too small, the screens are rapidly destroyed. Provision must be made for a suitable water supply to the interior of the mortar, the supply to be capable of very exact regulation. Roughly speaking an ordinary stamp battery requires about 100 cubic feet of water per hour for each mortar, but this figure is liable to great variation according to the rate of crushing and the nature of the material crushed. The mortar is generally fitted with a light cover through which the stamp stems pass.

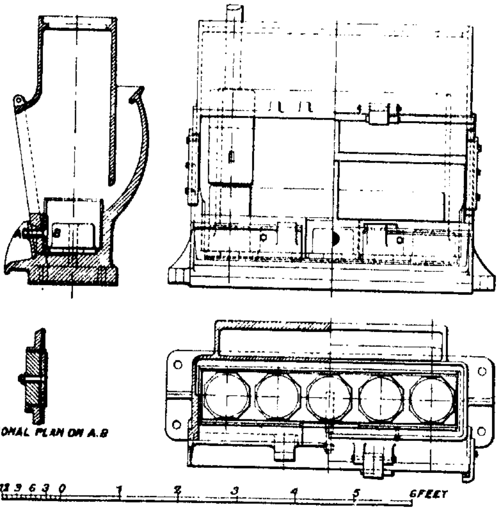


Fig. 141. Mortar. Vertical section, elevation and plan.

The dies (Fig. 142) are generally cylindrical with a square or octagonal base, the cylindrical portion being equal to the diameter of the stamp shoe that falls upon it, and 5 to 7 inches in depth. They are now mostly made of forged or cast steel.

The stamp proper consists of four separate parts, namely, the head or boss (Fig. 143), the stem (Fig. 144), the shoe (Fig. 145), and the tappet (Fig. 146), these being put together as shewn in Fig. 136. The crushing power of a stamp mill is determined mainly by the weight of the stamp, hence mills are usually designated by this figure; the weight may range from 400 to 1500 lbs., but in modern stamp mills the weight of the stamp is usually between 900 and 1200 lbs. The stamp stem consists of a bar of mild steel or best wrought iron, cold rolled or turned and with both ends slightly tapered; its length varies from 10 to 18 feet and its diameter from 2½ to 4 inches; its weight is generally between 40 and 45 per cent. of the total weight of the stamp. The head or boss is a cylinder of cast iron or in the best modern mills of cast steel, 16 to 20 inches high, and 8 to 10 inches in diameter, 9 inches being a usual figure for stamps weighing about 1000 lbs.; its weight is usually between 25 and 30 per cent. of the total falling weight. At the upper end it is

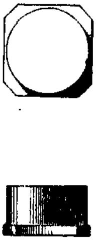


Fig. 142.
Die. Plan and elevation.

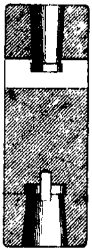


Fig. 143.
Head or Boss. Vertical section.

Fig. 144. Stamp-stem. Elevation.

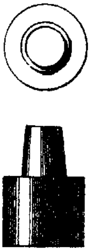


Fig. 145.
Shoe. Plan and elevation.

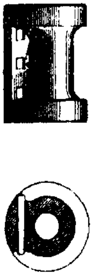
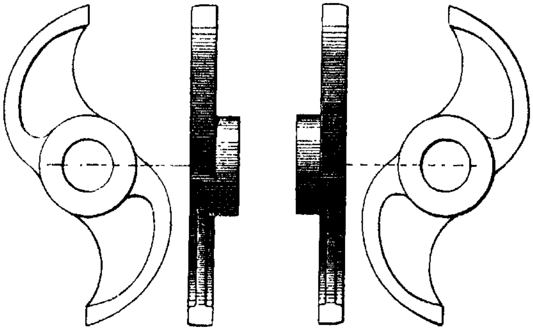


Fig. 146.
Tappet. Elevation and sectional plan.

bored out accurately to receive the tapered end of the stem, the lower end being similarly cored out (not bored as a rule) to receive the conical shank of the shoe; drift-ways are provided by means of which either the stem or the shoe may be driven out of the head when required. The shoe consists of a cylindrical butt, and of a conical shank by means of which it is attached to the head. The butt is of the same diameter as the head and between 5 and 7 inches deep; this the true wearing face of the entire stamp, and the deeper it is, the less often will the shoe have to be renewed, but on the other hand the greater will be the difference between the weights of a new and a worn-out shoe. The shoe weighs between 15 and 20 per cent. of the total falling weight, the butt weighing about ⅚ and the shank ⅙ of this amount. The shank is secured in the head by means of wedges made of dry, soft wood. The material of the shoes is now usually cast or forged steel, special steels like chrome steel or manganese steel being much used. It is found in practice that a special cast steel shoe working upon a forged steel die gives very satisfactory results. Chilled cast iron, at one time very largely employed, is now going out of use to a great extent. The wear of steel shoes and dies may be taken as about 0·5 to 0·7 lb. of metal per ton of hard quartz crushed, whilst good chilled cast iron will wear about twice as fast. The wear of the shoe is to that of the die in proportions varying from 3 : 2 to 2 : 1.

The tappet now very generally used consists, as shewn in Fig. 146, of a cylinder of cast iron or cast steel, bored out to fit the stem accurately. Within the cylinder is a recess in which is placed a steel gib which can be wedged tightly against the stem by driving up two, or often three, tapered keys; the pressure thus produced is sufficient to hold the tappet firmly in place upon the stem without slipping. The lower face of the tappet forms a circular collar, about 3 inches broad, against which the cam works, thus lifting the tappet and with it the stamp. At the same time the circular form of the tappet-face allows the entire stamp to revolve on its axis. The tappet usually makes up between 12 and 15 per cent. of the entire falling weight of the stamp. In an older form a screw thread was cut on the stem and the tappet formed practically a large nut, held in place by a lock nut above it; this so-called screw tappet is still found in Australia, as in Fig. 150, but has been superseded by the gib tappet everywhere else.

The lifting mechanism consists of a cam shaft, 5 to 7 inches in diameter, upon which are threaded the requisite number of cams. The latter are usually two-armed, as shewn in Fig. 147, the arms, accurately shaped, springing from a boss through which the cam shaft passes, and may be either right- or left-handed, as shewn, to suit the construction of the battery. The correct shape of the cam arms is a matter of considerable importance; as in the Cornish stamps their surface should form the normal involute of a circle, the radius of which is equal to the distance between the axes of the cam shaft and the stamp stem, whilst its centre coincides with that of the former. So much of this involute is taken as will give the desired lift, the point of the cam being merely somewhat flattened. It is a property of this involute that the lengths of the arc of the pitch circle traversed in a given time will be equal to the height of lift. Hence if the height of lift be h inches and the radius of the pitch circle r inches, the arm of the cam will move through an angle a° during the time of the lift such that



Left hand. Right hand.

Fig. 147. Cams, left- and right-handed. Elevation.

$$a^\circ = \frac{h}{\pi r} 180^\circ,$$

whence $$h = \frac{\pi r a}{180} \text{ and } r = \frac{180h}{\pi a}.$$

From these equations the involute required to lift the stamp through a given height for a given angular motion can be set out for any required stamp mill. Moreover, as already stated, it is a property of this involute that the portion of the surface in contact with the tappet will always be horizontal, so that the lift takes place in a truly vertical line[1].

The cam is secured to the cam shaft by means of one, or, more often, two large keys, the keyways running the full length of the shaft. This arrangement however makes the replacement of a broken cam a matter of considerable difficulty, and in most modern mills the arrangement known as the Blanton cam, Fig. 148, introduced by Messrs Fraser and Chalmers, Ltd., is employed. In this a curved wedge, A, Fig. 148, gripping tightly in a recess cut out of the boss of the cam is substituted for the keyway, a very powerful friction grip being thus produced, the curved wedge being held in place merely by a small set-screw.

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[1] For a full discussion of the geometrical construction and properties of this curve, see the Appendix to the Author's *Handbook of Gold-Milling.*

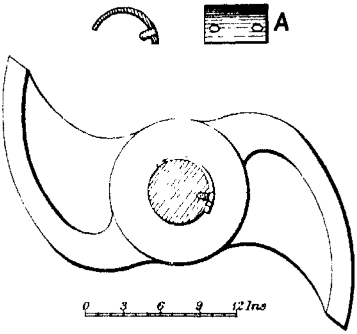

Fig 148. Blanton cam.

The order in which the stamps fall is not a matter of indifference; most makers of stamp mills observe the rule that neighbouring stamps shall never be allowed to fall in succession, whilst any order that tends to accumulate the ore to be crushed at one or other end of the mortar is most objectionable. In order to distribute the strain on the cam shaft equally it is important that the cams be uniformly distributed round the cam shaft, so that if s be the number of stamps lifted by one shaft, the angle between the cams lifting successive stamps must be $\frac{180^\circ}{s}$.

As the cam is two-armed, it is evident that each stamp makes two drops for each complete revolution of the cam shaft; the usual speed of modern stamp mills is between 85 and 110 drops per minute, the speed being within certain limits dependent on the length of drop, so that the former can only be increased by diminishing the latter. The length of drop ranges from 4 to 16 inches, but is usually under 8 inches. A very usual rate of working in modern mills is to run at 90 to 95 six-inch drops per minute. If the mineral be broken, as it should be, to less than

2 inch cubes before it is fed into the stamp mill, a 6 inch blow is ample to crush the hardest ore with stamps of 900 lbs. or more in weight.

The power required to drive a stamp mill is consumed wholly in lifting the stamp and is therefore the same whatever the crushing capacity of the mill may be, or whether the mill be crushing ore or not, as long as the conditions of driving remain unaltered. If n be the number of drops per minute, h the length of drop in inches, s the number of stamps, and w the weight of a stamp, the theoretical power required to drive the mill would be

$$\frac{nhsw}{33000 \times 12} \text{ H.P.}$$

If 25 to 30 per cent. be added to the figure so obtained, the result will be approximately correct; an exact formula will be found in the writer's *Handbook of Gold-milling*, already referred to.

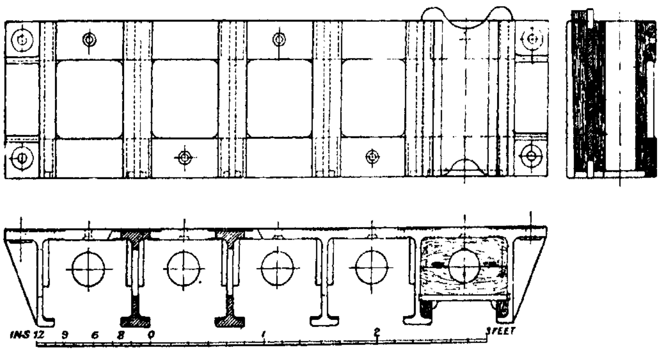

Fig. 149. Stamp guides.

The cam shaft is supported in bearings carried by the battery frame; this frame also carries the upper and lower sets of guides which keep the stamp stems truly vertical when working. Guides of many different forms are used; a good useful type, generally used in California, is shewn in Fig. 149; in this the guides proper are of wood, held in a cast-iron frame by means of wedges. The frames are of different designs according to circumstances; they must however always be very massive and substantially constructed as they are exposed to very heavy strains; they are generally made of wood, as shewn in Figs. 136—138, especially in America and also in South Africa, but steel, as in Fig. 139, wrought iron, and cast iron are also used.

The latter is a very satisfactory material, and is much used in Australia. The Australian design differs in many respects from the regular Californian type, and the details of a stamp battery, built by Messrs Thompson and Company, of Castlemaine, Victoria, are shewn in Fig. 150. It will be seen that the battery box is kept narrower, and that the screen in the front is vertical instead of inclining outwards at about 10°, as in the American pattern. The cam shaft in the Australian mills is usually driven by gearing off a main lay shaft, a clutch being used to throw out of or into action any battery as may be required, whilst in America belting is usually employed, heavy pulleys being keyed on to the cam shafts. A belt-tightener is often used, as in Fig. 138, for stopping or running any desired battery. In America the gib tappet, already described, is universal; in Australia, the screw tappet, shewn in Fig. 150, though inferior to the former in several respects, is, as before stated, still largely used.

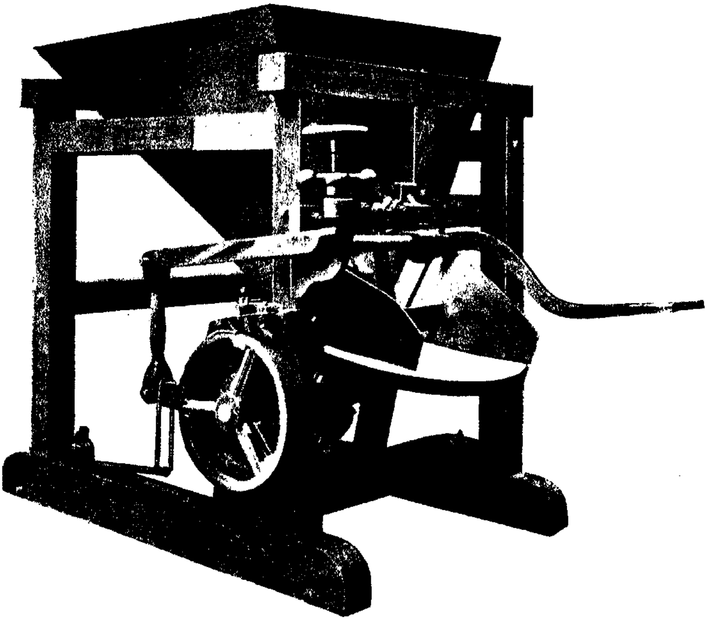

Fig. 151. Hendy challenge feeder. Perspective.

As it is a matter of great importance that the stamp mill should be regularly and uniformly supplied with the mineral to be crushed, because the consumption of power is the same whether it is crushing mineral or not, automatic ore-feeders of various kinds are very generally employed. These are usually worked by a tappet attached to the middle stamp, so that whenever the latter is on the point of touching the bare die, a fresh supply of ore is caused to drop into the mortar. The most successful is **Hendy's Challenge feeder**, shewn in Fig. 151, in which the blow of the tappet is communicated to the bar projecting in front, which carries a friction ratchet, which latter in turn causes an obliquely set plate, upon which the ore rests, to revolve through a small arc in such a

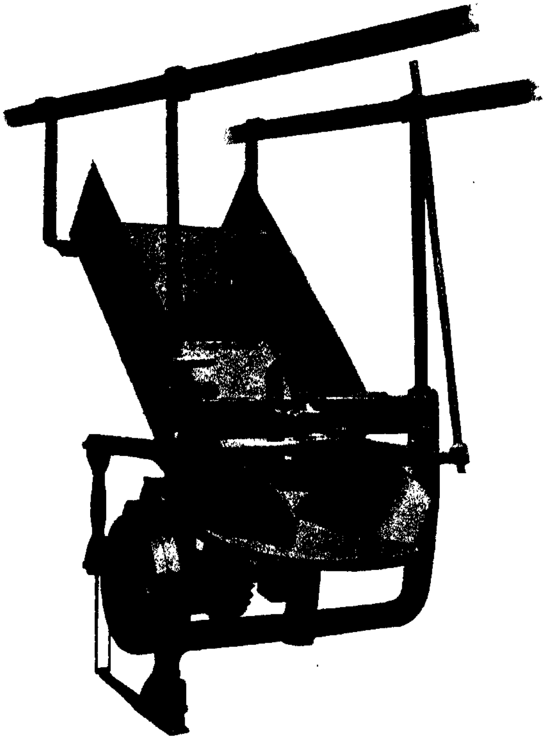

Fig. 152. Suspended Hendy challenge feeder. Perspective.

way as to sweep a certain amount of the ore it supports into the mortar, the amount so dropped in being proportional to the angular motion of the plate. Many other devices, more or less similar in design, are also employed, some of which are described in Chapter XI.

Fig. 152 shews a more convenient arrangement of the Challenge ore-feeder, known as the suspended Challenge ore-feeder, in which the appliance is suspended to a couple of light beams or girders beneath the shoot leading from the ore-bin.

The crushing capacity of a Californian stamp mill varies from 2 to 6 tons per head per 24 hours, 5 tons being a usual amount for a 1000 lb. stamp working on fairly hard rock. The cost of a Californian stamp mill varies between £50 and £100 per head according to its size and to what may be included in the specification.

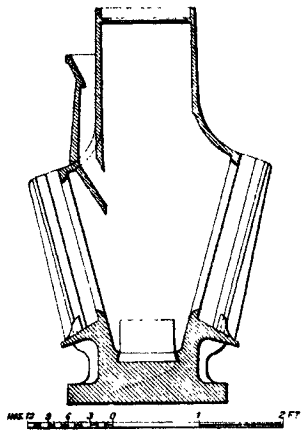

Fig. 153. Double discharge mortar. Vertical section.

The Californian stamp mill is occasionally used for dry crushing, for which it is however very poorly adapted. Much difficulty is experienced in getting the crushed material out of the mortar, whilst the shoes are apt to give trouble by working loose in the heads. When used for dry crushing, the cover of the mortar is made to fit as tight as possible, the mortar is generally supplied with a screen at the back as well as one along the front, as shewn in Fig. 153, and these screens are usually enclosed in a sheet iron casing in which runs a screw conveyor, or else which communicates with an exhaust fan by means of which the crushed ore is drawn out of the mortar box; the air current is made to traverse a series of settling chambers in which the crushed ore is deposited and finally passes through a filter consisting of bags made of sackcloth or some similar material, in which the fine dust is caught. Even with all these precautions much dust escapes into the mill building, where it not only causes loss, but rapidly destroys the bearings etc. of the machinery. The capacity of a dry crushing mill is moreover small, being usually only one-third of that of the same mill working wet.

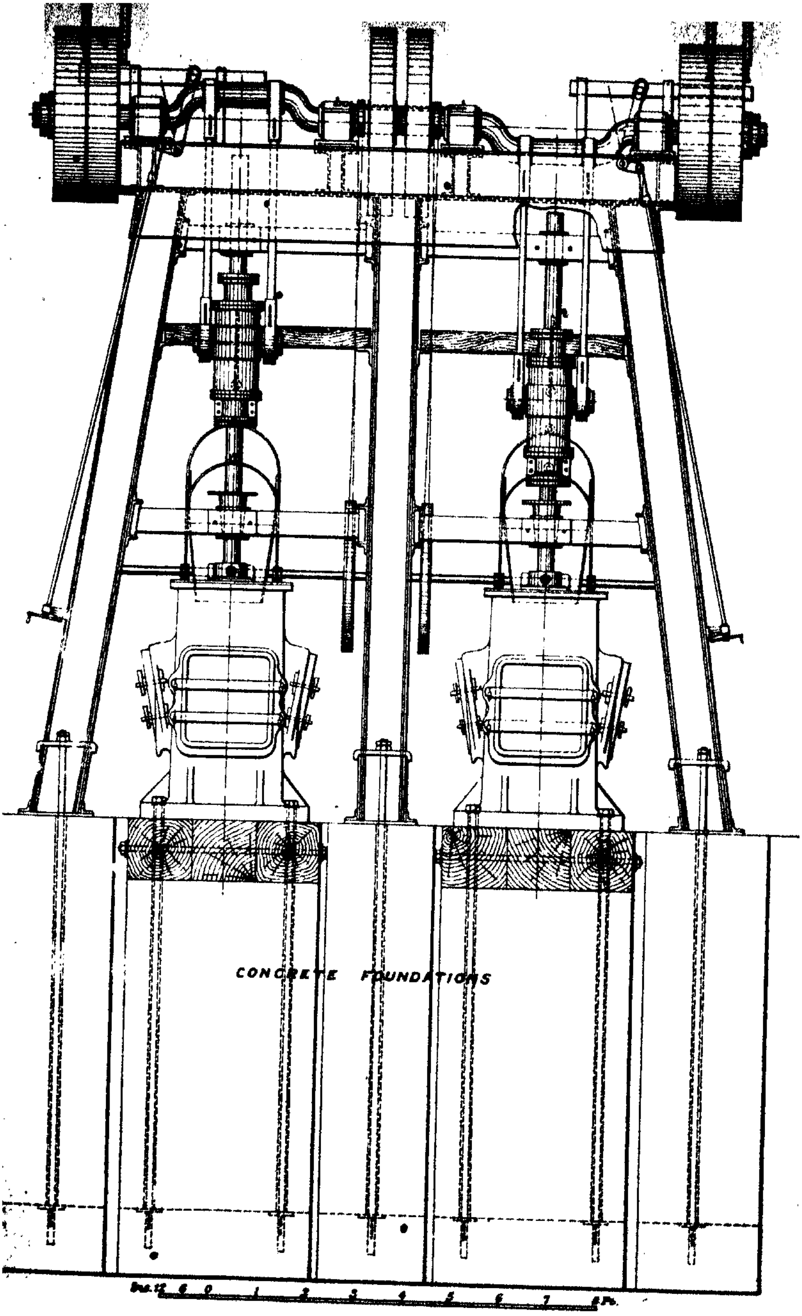

Fig. 154. Husband atmospheric stamp. Front elevation.

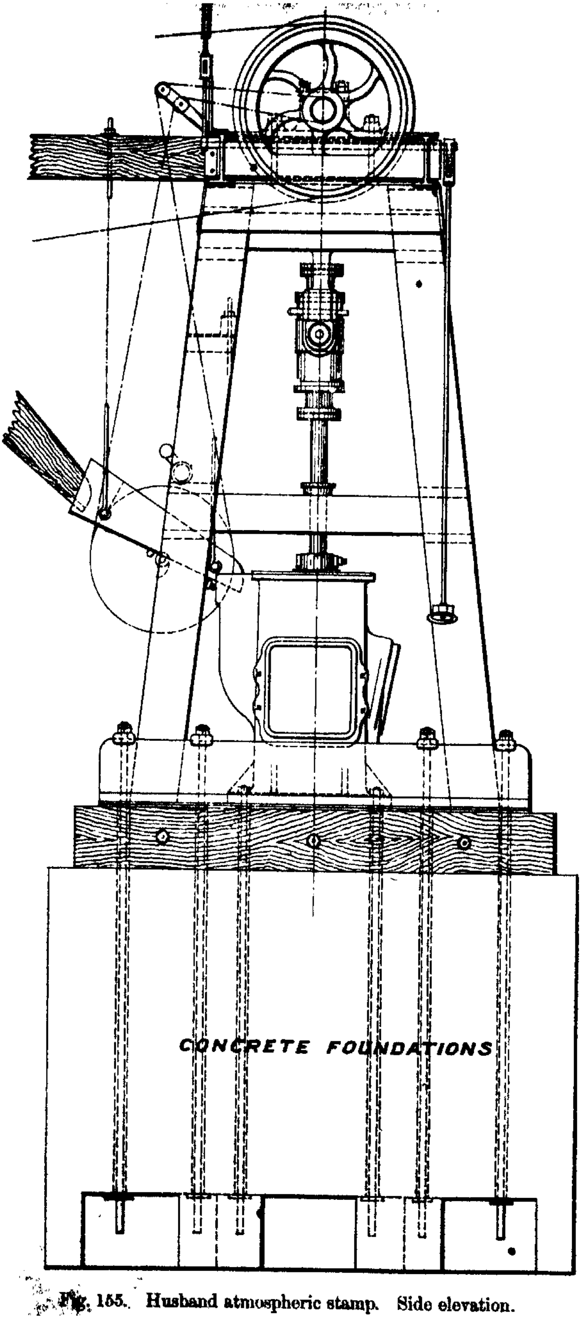

Fig. 155. Husband atmospheric stamp. Side elevation.

Power stamps. These are of two kinds, namely, such as act partially on the principle of the gravitation stamp, the downward motion of the stamp being merely accelerated by mechanical means, and those in which the crushing blow is due chiefly to the expansive force of steam; the latter may be conveniently described as steam stamps.

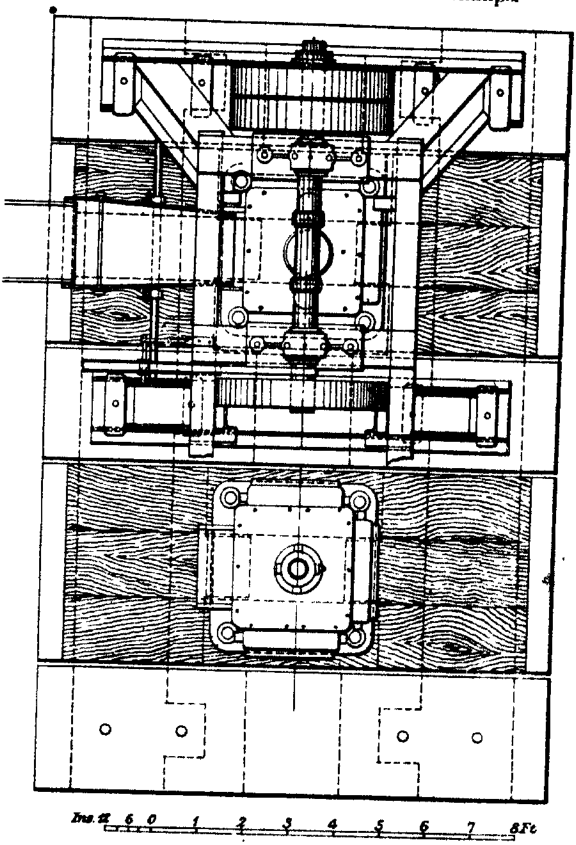

Fig. 156. Husband atmospheric stamp. Plan.

Numerous attempts have been made to accelerate the descent of the ordinary stamp by means of spiral springs placed above the tappets which are compressed whilst the stamp is being lifted by the cam; this method has proved ineffective mainly because the compressed spring exerts its greatest power at the time when this is least needed, namely at the commencement of the fall of the stamp. Another method, namely that of forcing the stamps downwards by means of a second cam shaft placed above the tappets, has been equally unsuccessful.

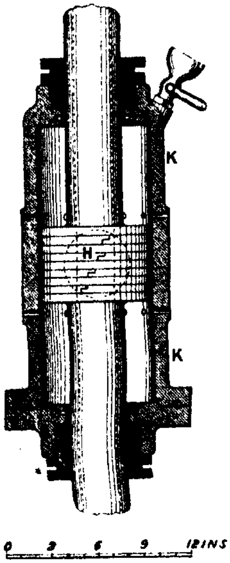

Fig. 157. Cylinder of Husband stamp. Vertical section.

An obvious method consists in actuating the stamp by means of a crank or its equivalent, so as to communicate power to it in its descent; this cannot be done by attaching the stamp directly to the crank or to a connecting rod actuated by it, because the shock of the falling stamp would destroy any ordinary machinery if communicated to it through rigid parts, whilst the varying depth of the layer of ore upon the die must also be taken into account. A spring of some kind must accordingly be interposed between the stamp head and the crank. Strong steel springs like railway waggon springs were used in a machine known as Patterson's Elephant stamp, which consisted of a pair of small square stamp heads working at a high speed in a small mortar box. The wear and tear was however found to be excessive and the machine was not a success. The Dunham stamp worked on somewhat the same principle, but has been equally unsuccessful.

In the **Husband atmospheric stamp,** the place of the spring is practically taken by air under pressure. The construction of this machine is shewn in Figs. 154, 155, 156. It consists of an iron framework supporting a crank shaft, there being two cranks upon it at an angle of 180° to each other. Each crank works a vertical cylinder by means of a forked connecting rod, attached to trunnions on the cylinders. As shewn in the section in Fig. 157, inside each cylinder *K*, there is a piston *H* to the lower end of which a stamp head is attached, and there are holes in the cylinder for the admission of air above and below the piston, when the latter is in the middle of the cylinder. As the cylinder moves upwards the piston immediately closes the lower apertures, and the air below it is compressed until it is under

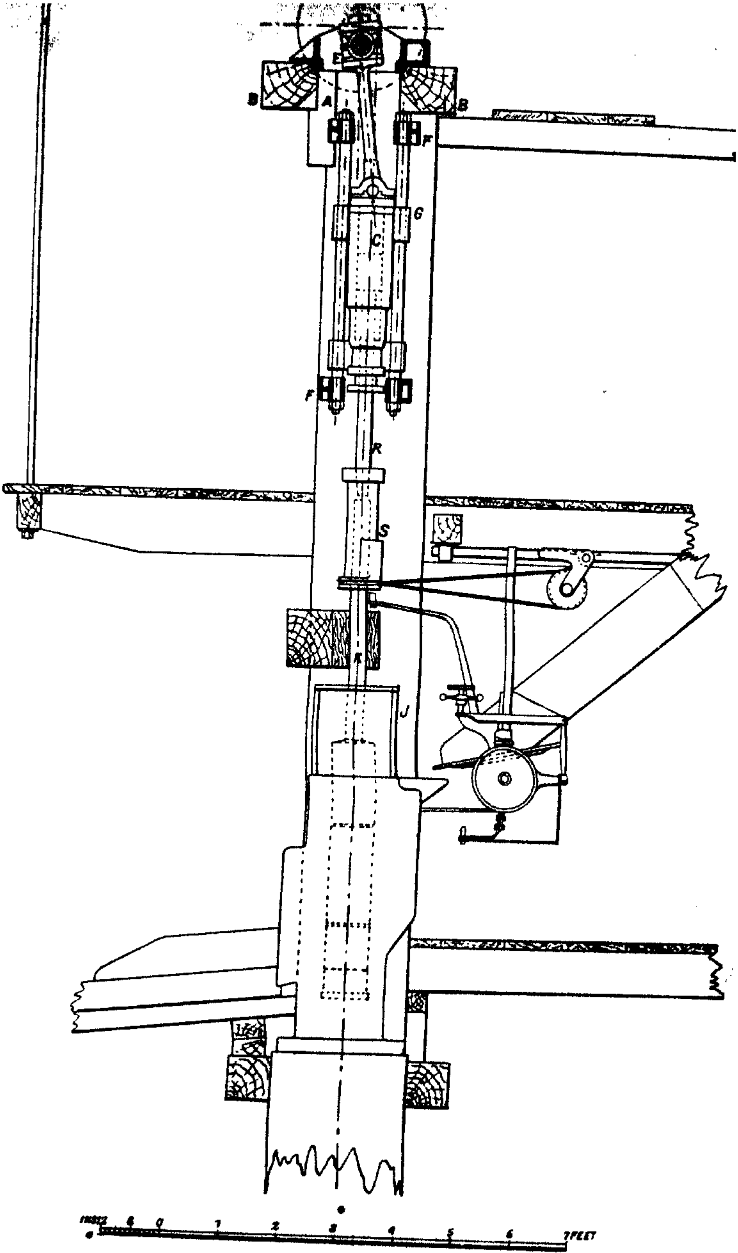

Fig. 158. Morison high-speed stamp. Sectional elevation.

a total pressure greater than the weight of the piston and stamp head, when the latter is lifted and continues to ascend at the same rate as the cylinder until the latter is in its highest position. The cylinder then commences to descend and the stamp to fall, but the former, moving more rapidly than the latter, soon overtakes it, and the air, now compressed above the piston, forces the latter down with a velocity equal to that of the cylinder itself. The air thus forms an elastic cushion at either end of the stroke that prevents any injurious shocks from being transmitted to the frame-work. As shewn, the two stamps work side by side, each in a cast iron mortar, which is supplied with screens on three sides, the fourth being occupied by the feed-shoot. The machine is strong, compact and does good work. One of these machines (with two stamp heads) is said to require 35 I.H.P. to drive it, and to be capable of crushing 40 tons of hard tinstone in 24 hours from 4 inch cubes to a mesh of 0·048 inch. A pair of these stamps stamped 10,760 tons of hard tinstuff in a year at Dolcoath, equal to 16½ tons per head per day. The water supply in this machine is through the hollow stamp stem, which is thus water-cooled. The Sholl pneumatic stamp is somewhat similar in principle, but the stamp stem is attached to the bottom of the cylinder whilst the piston rod which passes out through the top of the cylinder is worked by the crank. A Sholl mill running at 125 drops per minute and absorbing about 20 H.P. is said to be capable of crushing 24 tons to 14 mesh in 24 hours.

The most modern machine of this type is the **Morison High-speed stamp**[1], which has a hydro-pneumatic lifting cylinder. The stamp is shewn in elevation in Fig. 158 and the cylinder in section on a larger scale in Fig. 159. The mortar box *J*, stamp heads, etc., as also the frame and the lower guide are identical with those of an ordinary Californian stamp mill, the lifting mechanism alone being different, whilst the upper part of the frames is modified at *B*, so as to support this mechanism. The short stamp stem *K* (Fig. 158) fits into a sleeve *S* by which it is coupled to a piston rod *R*; the latter terminates in a piston which works inside a cylinder *C*; the latter is caused to move up and down within guides by means of a connecting rod coupled to a crank shaft *E*. Such a shaft is placed over the top of each five stamp battery, and has five cranks spaced at 72° apart. By this means the cylinder above each stamp is caused to rise and fall vertically at the desired speed. The upper part of this cylinder is surrounded by an annular chamber *W*,

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[1] *Inst. Min. Met.*, "A Development in Gravitation Stamp Mills," by Morison and Bremner, Vol. VIII. (January, 1900), p. 156.

Fig. 159, which acts as a water reservoir, communicating with the lower part of the cylinder by means of a port *P* and also with an air vessel at *A*; there is also free communication between the upper part of the cylinder *B* and the water reservoir. Telescopic pipes allow the water supply in the reservoir to be renewed, increased or decreased as may be required by means of a small circulating pump. The action is as follows: when the stamp is resting upon the die, and the cylinder in its lowest position, the port *P* is open and communicates with the lower part of the cylinder *C* which is full of water. As the cylinder rises it forces this water into the reservoir *W*, until the area of the port gradually diminishing, the pressure becomes sufficient to lift the piston *E* and with it the stamp. When the cylinder has reached the top of its stroke and is descending the piston is able to fall freely, its motion being accelerated partly by the friction of the rapidly descending cylinder and partly by the pressure of the air above the piston. By this means a much higher speed of driving can be obtained than with the ordinary Californian stamp. Suitable mechanism for turning the stamp slightly after each blow is provided. The inventor considers 132 blows per minute to be the best rate of working; and that his 1400 lb. stamp working at this rate has a crushing capacity 70 per cent. greater than that of an 1150 lb. Californian stamp, or in other words that 30 heads of such high speed stamps are equivalent to 50 heads of Californian stamps. He does not however claim any marked power economy for his stamp; it must be added that the machine is new and has not come into use to any extent.

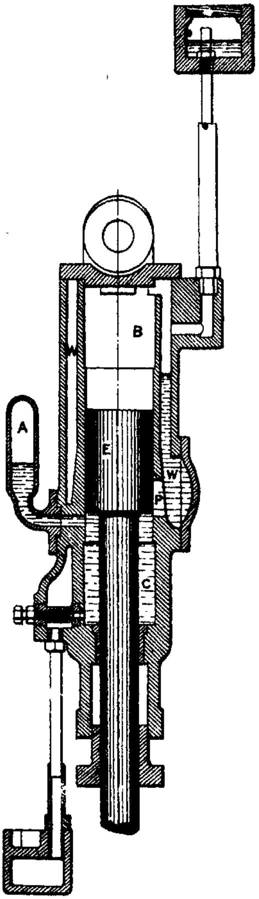

Fig. 159. Cylinder of Morison high-speed stamp. Vertical section.

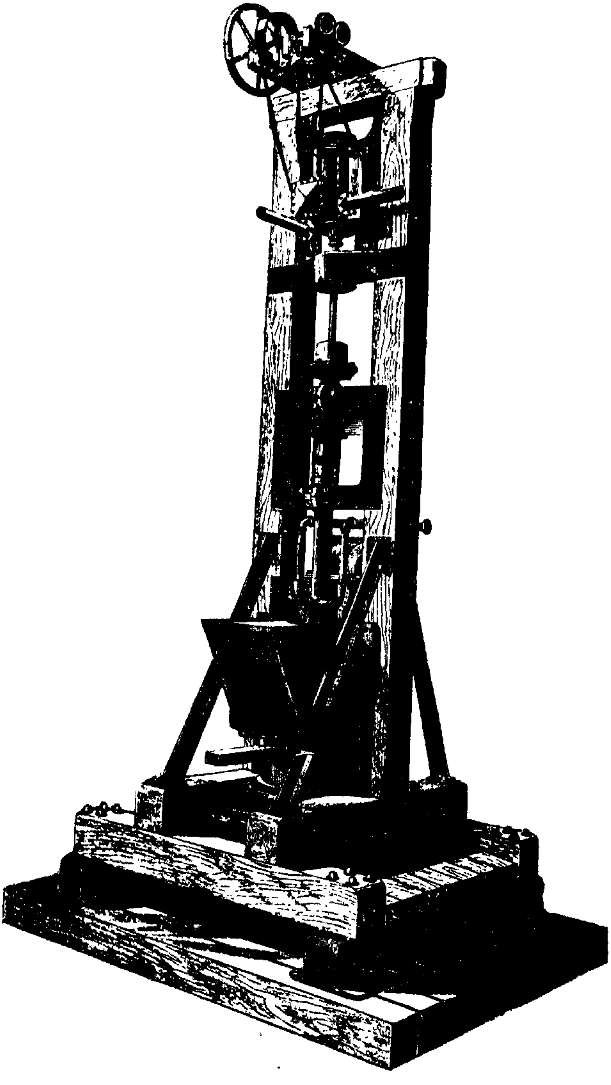

Fig. 160. Ball steam stamp. Perspective.

Steam stamps have been devised of many different forms, the application of the principle of the steam hammer to crushing purposes being a fairly obvious one. By far the most important development of the steam stamp has been for the purpose of crushing the copper bearing rock of the famous Lake Superior copper mines; this material does not need very fine crushing, the screens employed being punched with holes about $\frac{3}{16}$ inch or $\frac{1}{4}$ inch in diameter. The first of these

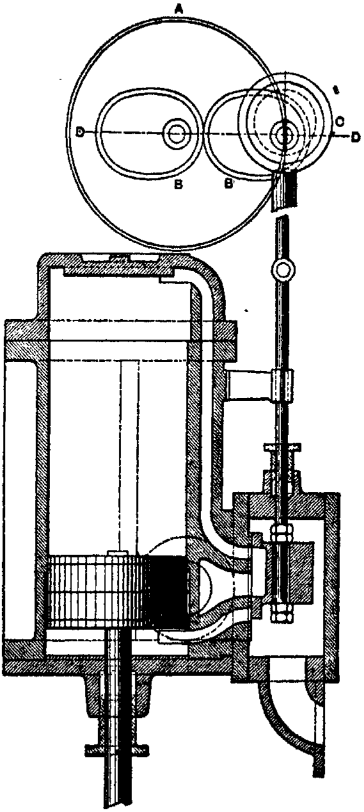

Fig. 161. Cylinder of Ball steam stamp. Vertical section.

steam stamps is said to have been designed by Mr W. Ball in 1856[1]; and the original design with but slight modifications is still in use in many places. A modern type of Ball steam stamp is shewn in Fig. 160. The mortar rests upon a massive iron bedplate, which is in its turn supported upon sills of wooden and iron beams arranged crosswise so as to give a certain amount of elasticity; the total weight of this anvil is about 11 tons. Strong wooden uprights carry the steam cylinder, which has a diameter of about 15 inches and a stroke of 24 inches. The valve gear consists of a plain D-slide valve, driven by a small eccentric keyed to a shaft carried on top of the uprights and independently driven. The steam cylinder and valve gear are shewn in Fig. 161. The eccentric *C*, which is connected direct to the stem of the D-slide valve, is itself worked by an eccentric gear, consisting of a pair of elliptical spur wheels *B*, *B′*, driven by the pulley *A*, the throw of the eccentric being at right angles to the line *D*, *D*, in which the major axes of the ellipses lie. This arrangement opens the port fully for the downstroke, but only very little—about $\frac{3}{16}$ inch—for the upstroke. The stamp stem is 8 inches in diameter, to which the shoe, rectangular in plan, is attached by means of a dovetail and key. The weight of the falling parts is about 4,500 lbs. and it works at 90 drops per minute. The crushing capacity of this stamp is about 150 tons in 24 hours. Large clearance spaces must be left, both at the top and bottom of the cylinder, so that much steam is wasted and the machine is by no means an economical one. The Leavitt steam stamp shewn in Fig. 162 was a considerable improvement in this respect. It will be seen that the shape of the mortar is practically unchanged; the frame now consists of four cast iron columns, but the chief modifications are in the cylinder. This, shewn on a larger scale in Fig. 163, is steam-jacketed throughout, and is differential, the lower portion having the smaller diameter. The upper cylinder is 21½ inches in diameter and the lower 14 inches. A steam dashpot is formed by the lower end of the lower cylinder, an air dashpot being used to cushion the stamp and prevent its rising too high. The valve gear (not shewn in the figure) consists of four cams, each working against a lever furnished with a roller so as to diminish friction. The object of the differential cylinder is to economise steam, only enough being used in the lower cylinder to just lift the weight of the piston and the attached stamp head. The mortar was made heavier in the most

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[1] *Amer. Inst. Mech. Eng.*, "Notes on the Steam Stamp," by F. G. Coggin, Vol. VI. 1885 p 370

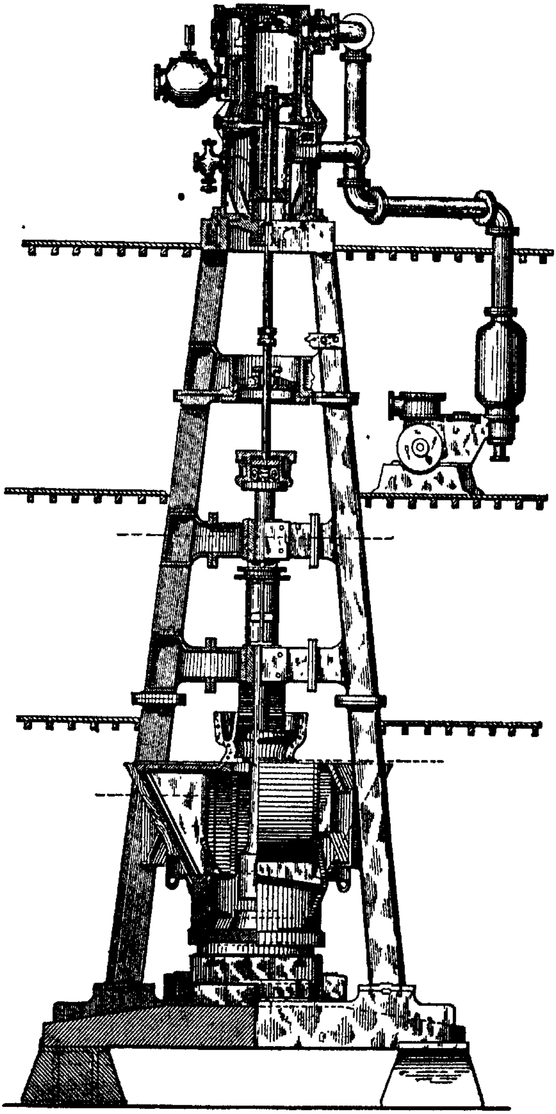

Fig. 162. Leavitt steam stamp. Elevation, partly in section.

recent Leavitt stamps, and the weight of the anvil was increased to 12 tons. It is stated that the Leavitt stamp saved 35 to 40 per cent of fuel as compared with the Ball stamp, whilst its crushing capacity was increased to 240 tons in 24 hours. The chilled cast iron shoe wears for about 6 days. A set of steel screens $\frac{1}{10}$ inch thick, punched with $\frac{3}{16}$ inch holes, will screen about 10,000 tons of crushed rock. The water consumption in the mortar is about 800 cubic feet to the ton of rock.

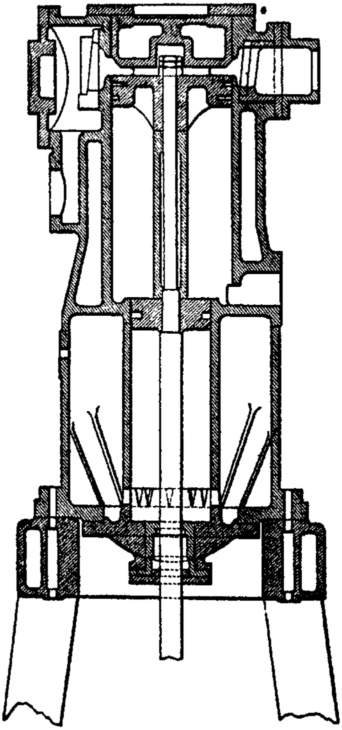

Fig. 163. Cylinder of Leavitt steam stamp. Vertical section.

The Allis steam stamp is a development of the last-named, its rushing capacity being considerably increased. All these steam stamps re stated however to give about the same efficiency, namely from 1·745 to 1·852 tons per H.P. in 24 hours.

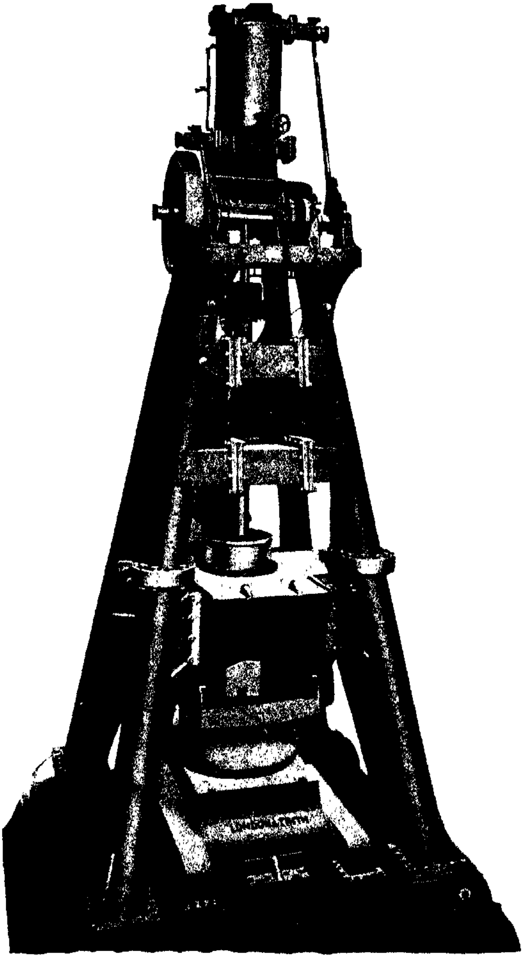

Fig. 164. Allis steam stamp. Perspective.

The Allis stamp is shewn in Fig. 164 as built by Messrs Fraser and Chalmers, Ltd. The valve gear has a special quick return motion, giving very little steam on the upstroke, and a full head of steam with an early cut-off on the down-stroke. The stamp shoe is of chilled iron, and the mortar is lined with chilled iron plates and rests on a cast iron base plate weighing 12 tons underneath each mortar. One of these stamps, running under a steam pressure of 90 lbs. per sq. inch at 100 drops per minute, crushes on an average 360 tons per day of amygdaloid through a $\frac{3}{16}$ inch screen, and uses 500,000 gallons of water per day; about $1\frac{1}{4}$ hours out of every 24 are taken up with removing the lump copper from the mortar. With a steam pressure of 115 lbs. per sq. inch, the same speed of stamp, and the same size of screen perforations, 450 tons have been crushed in 23 hours. In this mill 28 tons of rock are crushed per ton of coal consumed, this serving also for heating, lighting and pumping. Such a steam stamp weighs (without the cast iron base block) about 130 tons and costs about £2,800.

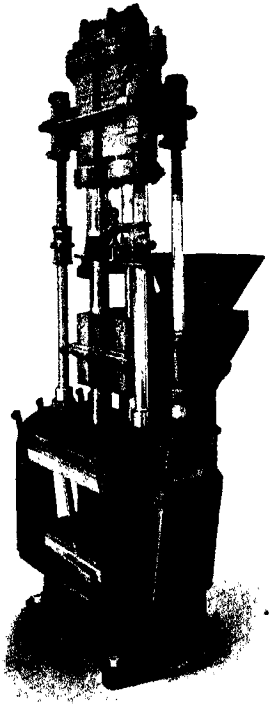

Fig. 165.
Tremain steam stamp. Perspective.

Numerous forms of steam stamps have been devised for fine crushing; perhaps the best known and the only one that need be described is the **Tremain steam stamp**[1] shewn in perspective in Fig. 165 and in sectional elevation in Fig. 166. This machine consists of a pair of stamps working in a mortar furnished with screens on three sides. The stamp consists of a stem with a piston at the upper and a shoe at the lower end; there is no head strictly speaking, the shoe fitting on to the stem as shewn. The stem is 4 inches

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[1] *Trans. Amer. Inst. Min. Eng.*, "The Use of the Tremain Steam Stamp,' by E. A. Sperry. Vol. xxvi. 1896. p. 545.

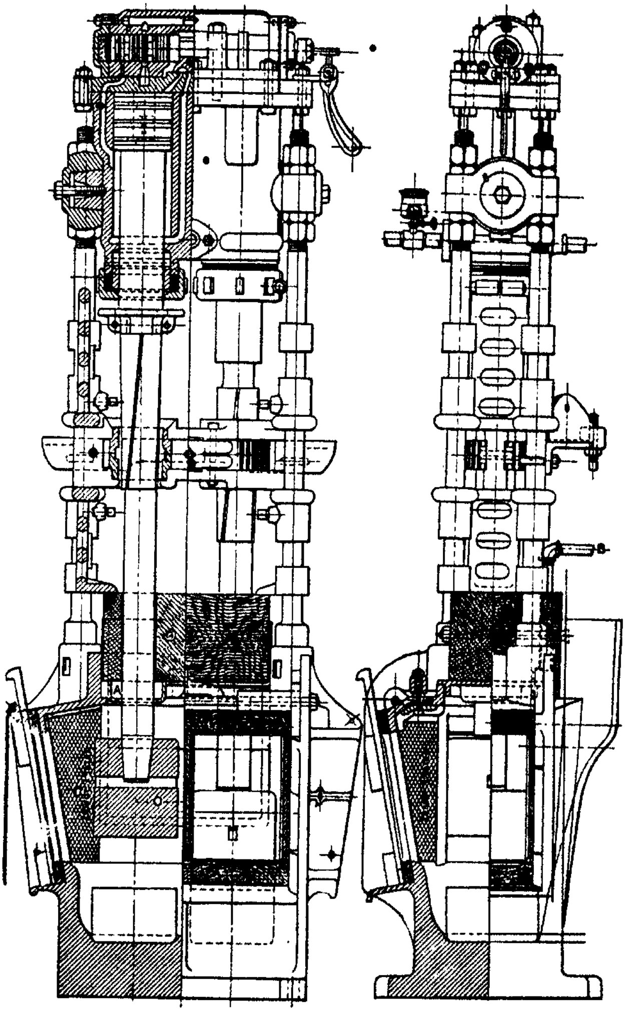

Fig. 166. Tremain steam stamp. Sectional front and side elevations.

in diameter and the piston 6 inches; there is therefore only an annular piston, 1 inch wide, exposed to the action of the steam in order to lift the stamp. The falling weight is about 300 lbs., the length of drop is from 5 to 8 inches, and the stamp is capable of working up to 200 drops per minute. The average capacity of the machine was found to be about 12 tons in 24 hours, crushed from 3 inch cube down to 20 mesh. The fuel used amounted to about 0·2 cord of wood per ton of ore, equivalent to about 0·1 ton of ordinary coal. The makers have devised a special form of ore-feeder to be used with this mill. The stamp mill alone weighs about 1½ tons and costs about £170; the mill, with a nominal 12 H.P. boiler and all fittings complete, weighs about 3½ tons, and costs about £250.

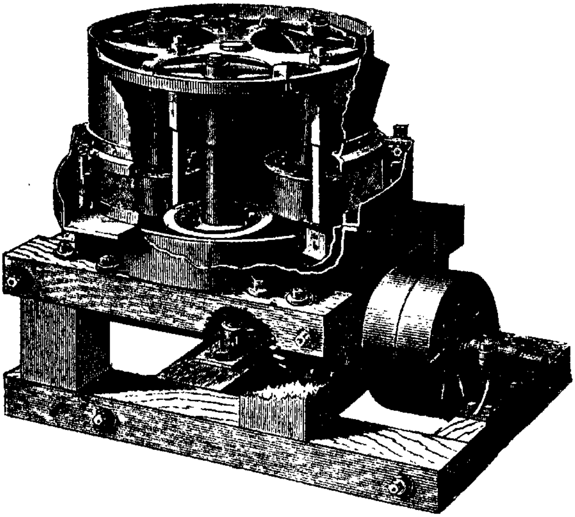

Fig. 167. Huntingdon mill. Sectional perspective.

Rotating fine crushers. A number of fine crushing machines act chiefly by the percussive effect of weights against a revolving track, the force impelling the weights being either centrifugal force or gravity; in all these machines abrasion, and to a smaller extent direct pressure, may also take some part. The weights take the shape either of balls or of rollers, and they work against an anvil or track which revolves about a vertical axis when it is mainly centrifugal force that comes into play, and about a horizontal axis when gravity is taken advantage of. These machines play a comparatively subordinate part in general crushing operations; the **Huntingdon mill** may be taken as a good typical example. It is shewn in sectional perspective in Fig. 167 and in vertical section in Fig. 168. It consists essentially of an iron drum, the lower part of which carries a steel ring which forms the track against which the crushing is performed. Immediately above this ring are placed three screens, about 9 inches deep, which occupy the front half of the circumference of the drum. Below these screens is a semicircular launder which receives the pulp discharged through the screens. In the centre of the drum rises a tapered iron sleeve, through which passes a vertical shaft carried upon a step bearing and rotated by means of suitable gearing.

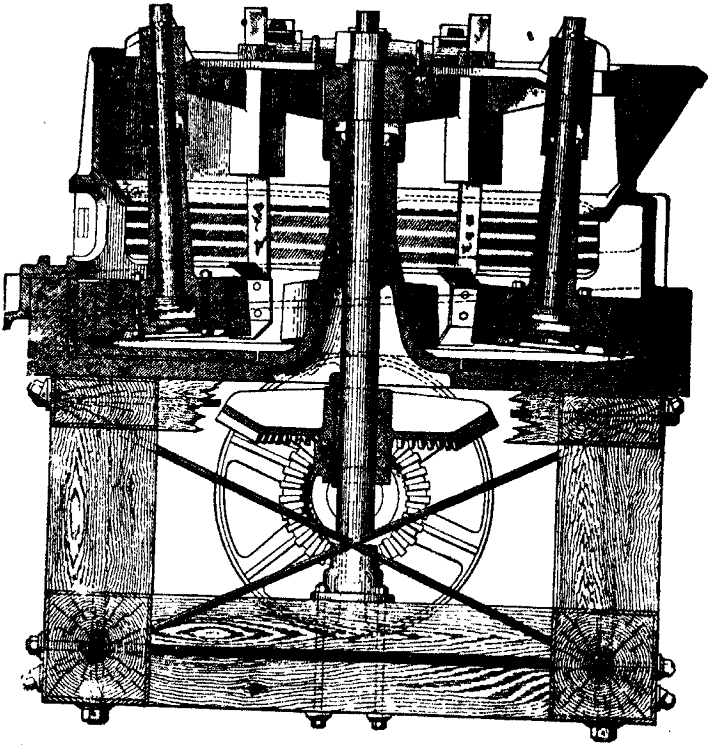

Fig. 168. Huntingdon mill. Vertical section.

To the top of the shaft is keyed a cross-shaped casting, from which four spindles are suspended by means of yokes; the lower end of each of these spindles is provided with a steel roller which works against the above-mentioned annular track. These rollers are capable of revolving on the spindles, and the latter are overhung as shewn, so as to be pressed outwards both by gravity and by centrifugal force against the steel ring. The material to be crushed is introduced through the feed shoot at the back of the machine.

The Huntingdon mill is made in three sizes, namely:

	Diameter	Weight	Revolutions of spindle per minute	Approximate price
I.	3′ 6″	2 ton 18 cwt.	90	£175
II.	5′	4 ton 18 cwt.	70	£300
III.	6′	8 ton 18 cwt.	55	£470

The most usual size is the No. II, which seems to require from 10 to 12 H.P. to run it, the usual speed being only about 60 revolutions per minute, instead of the higher figure recommended by the makers. Its efficiency on hard material is small, being only about 10 tons per 24 hours crushed to about 15 mesh; on softer material it has been known to crush over 20 tons per 24 hours to 30 mesh (0·024 inch). The wear of the ring die and crushing rollers is considerable, amounting to about 10*d.* per ton of hard ore crushed. This mill is best suited for crushing moderately soft material, which should be broken to at the most 1 inch cube before being charged. It is also well adapted to the further crushing of once-crushed ore (as in the treatment of so-called "middlings" from jigs [see p. 228]). It has been used a good deal for the crushing and amalgamation of gold quartz, but is not so well adapted to this purpose, especially if the ore is dense and hard. This mill is only suitable for wet crushing.

This machine is a modification of the Dingey pulveriser, which was in use about the year 1870, and consisted also of a horizontal drum, 6 ft. in internal diameter, fitted with screens, and containing 4 rollers, which were rotated by gearing from a central spur-wheel; these rollers were 2 ft. 6 ins. in diameter and made 200 revolutions per minute round fixed axes, whilst the drum itself made 4 or 5 revolutions per minute. It is said to have been capable of crushing 15 to 20 tons per day of 24 hours.

The Howland* pulveriser was also an earlier machine of somewhat similar type to the Huntingdon mill, the crushing rollers being however smaller, more numerous, and not attached to spindles.

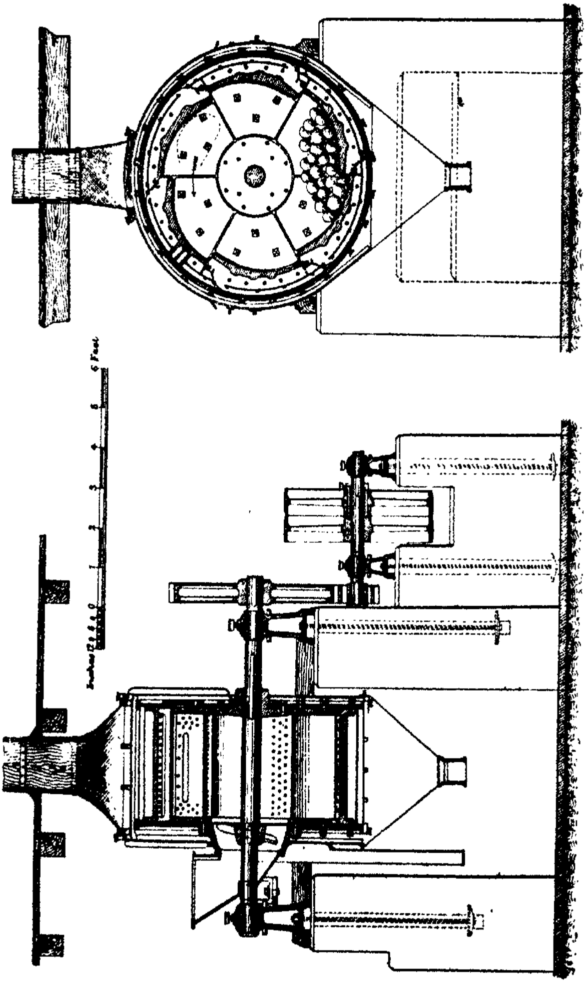

Fig. 169. Ball mill. Longitudinal and transverse sections.

The Griffin mill is like the Huntingdon mill with a single large roller, which is caused to revolve on its spindle by means of gearing.

Several machines, such as the "Cyclops" mill and the "Niagara" pulveriser, consist of a drum revolving about a horizontal axis, in which crushing is performed by balls or rollers working against an annular steel track. These machines have not been very successful in practice.

A very efficient machine, and one that is coming into general use, is the **Ball mill**, originally introduced by the Fried. Krupp A. G. Grusonwerk of Magdeburg, shewn in Fig. 169. It consists of a drum revolving about a horizontal axis, the circumference of which consists of screens of any desired mesh; within this is a coarser screen made of perforated bent steel plates, so arranged as to protect the outer fine screen, whilst at the same time allowing all the material that is not fine enough to get back into the interior of the machine. Inside the protecting screens are six strong chilled iron "hunch plates," helical in form. and arranged as shewn, so as to form steps. Upon these plates lie a number of steel balls. The material to be crushed is introduced through the feed shoot, shewn in the section, into the central portion of the drum; as the latter revolves the steel balls fall over each other at each step of the chilled hunch plates, and by their impact crush the mineral, partly between the balls themselves, partly between the balls and the hunch plates. The mill is completely enclosed, as shewn, in a sheet iron casing. It is designed for working dry, and is one of the most efficient of dry fine crushers, but can also be modified so as to work wet. The following table shews the principal sizes:

Nos.	01	1	2	3	4	5
iameter of mill (inches) ……	35	42	52	59	72	86
Vidth of mill (inches) …………	20	28	39	39	39	46
levolutions per minute ………	38	35	30	27	25	21
Iorse-power required ………	0·5	1·5	3	6	11	15
Veight of machine alone (cwt.)	21	40	66	87	120	182
Veight of a set of balls (cwt.)…	2	8	6	9	13	18
iameter of balls (inches) ……	2·4 & 3·2	3·2 & 4·0	4, 4·5 & 5	4, 4·5 & 5	4, 4·5 & 5	4, 4·5 & 5
rice, about, £ …………………	70	100	170	225	275	360

The capacity naturally varies very widely with the material to be crushed and the desired degree of fineness; moist mineral takes far longer to crush than dry, owing to the difficulty of passing moist crushed material through the screens.

Exhaustive records have been published by Mr White, based on the running of a set of sixteen No. 5 Ball mills at Mount Morgan on moderately hard quartzose ore. These shew that the Ball mills alone require

13 H.P. each and crush at the rate of 22·7 tons of ore in 24 hours to 20 mesh (0·025 inch), being at the rate of 1·75 tons per H.P. for 24 hours. The wear of steel is at the rate of 1·644 lbs. per ton of ore crushed, out of which the wear of the balls is 0·725 lb. and of hunch plates 0·681 lb.; the total cost of materials for renewals is given as 10·5*d.* per ton of ore crushed.

Similar machines are made by many other makers; Messrs Bowes, Scott and Western, Ltd. make a ball mill somewhat like the one described above, except that the material to be reduced is fed in through a hollow axis; the screens are attached in convenient segments so as to be easily removed and replaced. This mill is shewn in Fig. 170.

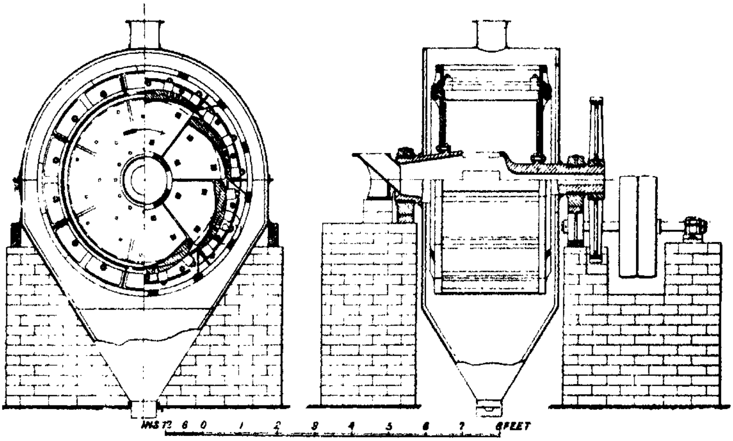

Fig. 170. Ball mill. Front and side elevations; partly in section.

Several ball mills have also been designed for wet crushing; one of the most efficient is the **Gröndal** ball mill, shewn in perspective in Fig. 171 and in section in Fig. 172. This mill is usually made 70 inches to 80 inches in inside diameter by 40 inches internal length; the cylindrical portion is built up of longitudinal steel bars, which can be worn down to less than half their thickness before they need to be discarded. These are fastened into massive cast iron end plates with renewable liners, and provided with central necks which form bearings, on which the whole drum revolves, although in some forms a central shaft has also been employed, and in others friction rollers have been used. The material to be crushed is fed in through one of the necks and escapes through the other, the degree of fineness to which material is crushed depending primarily upon the rate of flow of the water. In other forms a short conical screen is inserted in the discharge end. The mill is charged with about 2 tons of balls of chilled cast iron, 3 to 6 inches in diameter; as these wear

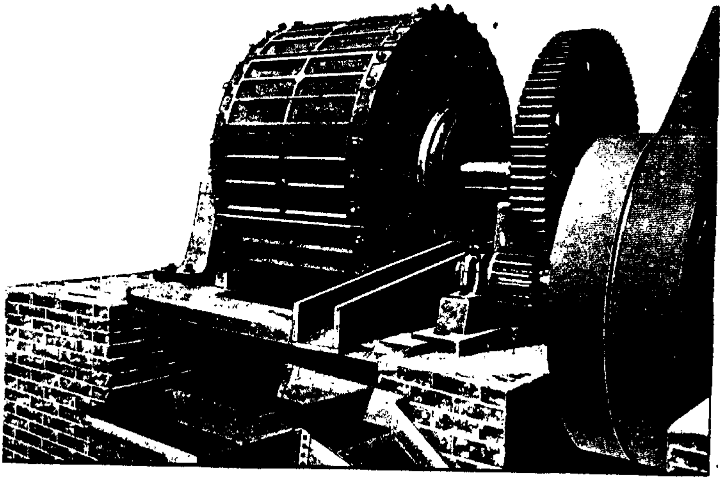

Fig. 171. Wet crushing Gröndal Ball mill. Perspective.

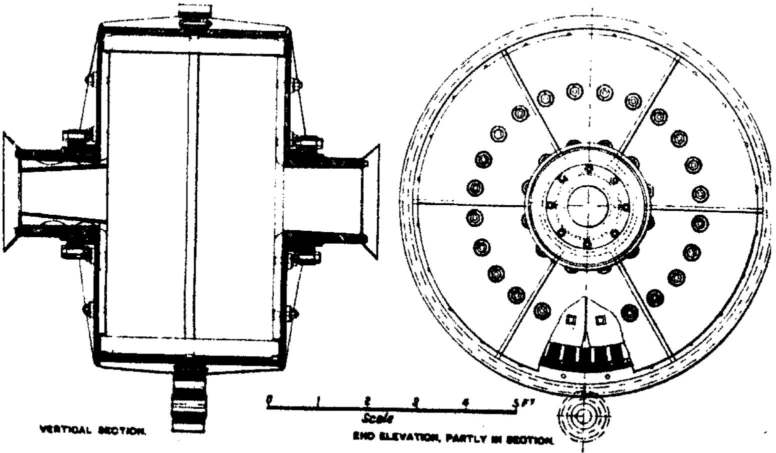

Fig. 172. Wet crushing Gröndal Ball mill. Vertical section and end elevation.

down others are regularly put in, so that in work the mill contains balls of all sizes from 6 inch downwards. The average wear of the balls amounts to about 2 lbs. of metal per ton of hard ore crushed, and about 3 lbs. of metal off the lining of the mill. These ball mills are run at about 25 to 28 revolutions per minute, require 25 to 35 H.P., and will crush 25 to 100 tons of hard ore down to 0·02 mesh in 24 hours; at this rate of working the water consumption is about 10 cubic feet per minute. These mills are used very extensively in Scandinavia for the wet fine crushing of hard iron ores. The price is about £350.

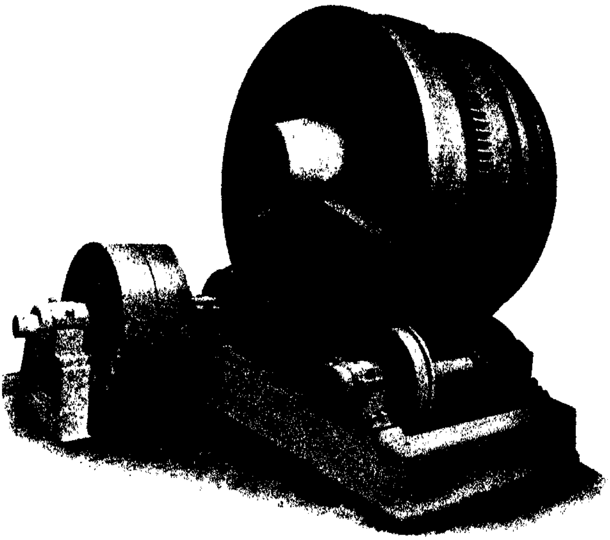

Fig. 173. Ferraris Ball mill. Perspective.

The **Ferraris** wet crushing ball mill is shewn in Fig. 173, which clearly indicates the mode of supporting and driving it. It is about 61½ inches in diameter and 40 inches long, divided longitudinally into two compartments; the longer, about 30 inches in length, is lined with manganese steel plates, and contains about ½ ton of forged steel balls, 4 inches and

6 inches in diameter. The shorter compartment is separated from the other by a transverse perforated partition; it carries a central cone placed axially with divisions extending radially from it, whilst the outer surface consists of a screen enclosed in a housing. The material is fed in through the axis into the longer compartment, and when crushed passes through the holes in the partition into the screening compartment; the undersize escapes through the screen, whilst the oversize, elevated in the radial divisions (which act like a raff wheel), is returned over the outside of the cone into the crushing compartment. This mill runs at 20 revolutions per minute and requires about 6 H.P. to drive it. It is said to be capable of crushing 20 to 35 tons per 24 hours, according as the screen varies from 30 to 12 mesh. This mill, which is made by Messrs Edgar, Allen and Co., is said to do very good work; its weight complete is 7½ tons and its price £350.

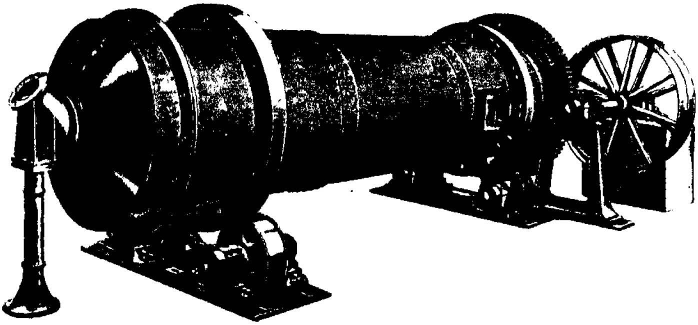

Fig. 174. Tube mill. Perspective.

For very fine crushing, wet or dry, such as is required in special cases, the **Tube mill** is employed, which appears to have been designed originally by Mr Davidsen. This is simply a very long ball mill; the balls may be either iron balls or else rounded natural flints may be used; in the same way the lining is either of metal (chilled cast iron, manganese steel, etc.) or else of pieces of hard flint or chert. The general appearance of such a mill as made by the Humboldt Engineering Co. is shewn in Fig. 174, and the construction of a similar one made by Fried. Krupp A. G. Grusonwerk in Fig. 175. The latter mill is about 4 feet diameter and 16 feet 6 inches long, and makes 29 revolutions

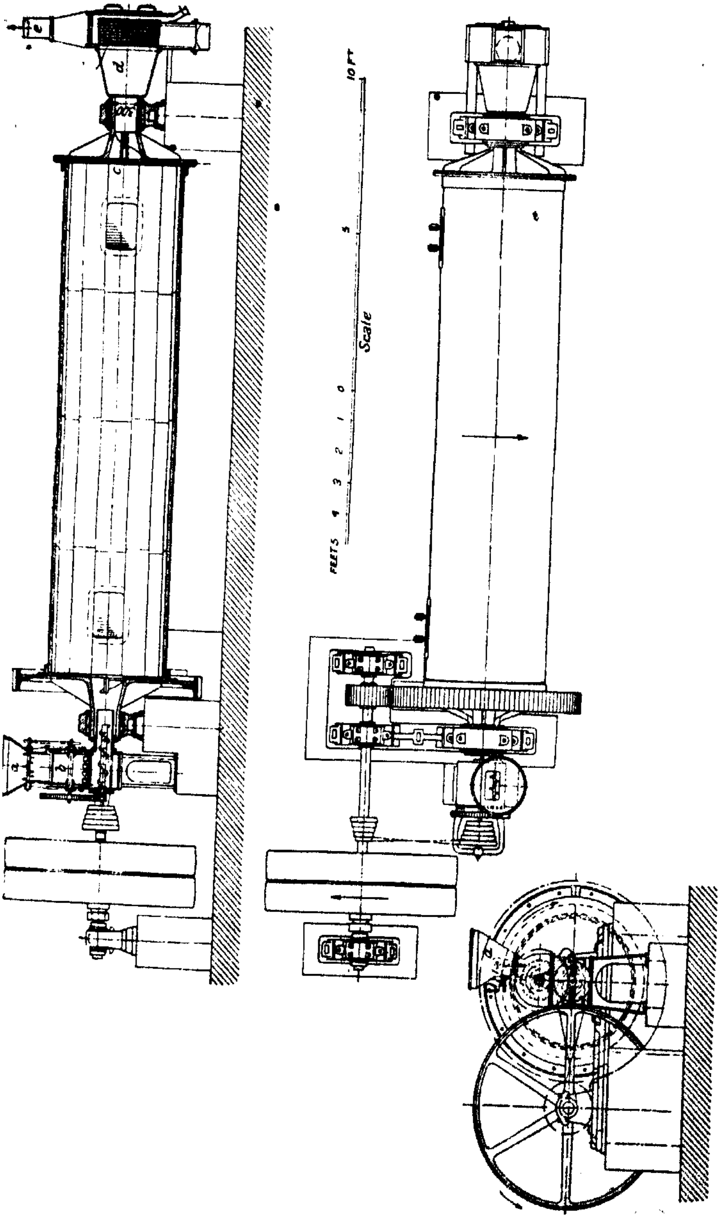

Fig. 175. Tube mill. Sectional elevation, plan and end elevation.

per minute. The material to be crushed is fed in through the hopper *a*, whence it passes into a feeding appliance *b*, and thence into the tube mill proper; at the opposite end it is discharged through a screen *c* and drops into the discharge hopper *d*. At *e* there is an exhaust pipe for carrying off fine dust. Tube mills vary in diameter from 3 feet to 5 feet, and in length from 13 to 16 feet. In other patterns the discharge, instead of being axial, is through screens round the periphery at the discharge end of the tube; both arrangements have their advocates. The Tube mill was originally intended for dry crushing, but is also used for crushing wet.

A mill of the size shewn above was found in West Australia[1] to be capable of crushing wet 38 tons of sands from a ball mill in 24 hours, 95 per cent. of the product being finer than 100 mesh. The power required was 30 H.P., and the costs per ton of sands are given as follows:

Power	11·53 pence
Flints and liners	1·85
Labour	3·50
Repairs	0·99
Total per ton	17·87 pence

In another mine in the same district a tube mill ground 67 tons of pulp from a stamp mill with 25 mesh screen down to 220 mesh in 24 hours, with a power consumption of 30 H.P. A set of hard iron liner plates wore out in 145 days, corresponding to 10,530 tons of ore, whilst the wear of the flint balls was 848 lbs. per 100 tons of ore. The preference here seemed to be for hard iron liners and flint balls.

Mr Davidsen states[2] that the best speed of revolution of the tube mill is $\frac{200}{\sqrt{D}}$, where D is the interior diameter in inches; the best charge of flint is given by the formula $W=44N$, where W is the weight of flints put into the mill, and N the internal capacity of the tube mill in cubic feet. In West Australia the actual practice corresponds more nearly to $W=60N$. The flints used should be natural pebbles, from ¾ inch to 4 inches in diameter, and are found on the shores of Denmark and the north coast of France, the latter being the better. The horse-power absorbed by a tube mill in normal work may be calculated as equal to $0{\cdot}15N$; it is said that the mill takes more power grinding dry than when grinding wet.

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[1] *Trans. Inst. Min. Met.* Vol. XIV. 1904–5, p. 87.

[2] *Ibid.* p. 154.

The theory of ball mills and tube mills (which are in effect merely prolonged ball mills, unless when the former are fitted, like the Gruson ball mill, with curved baffle plates) has been carefully worked out by Professor Hermann Fischer[1]. He has observed the action of the balls in specially made mills, and deduces that the action is as shewn in Fig. 176, which represents the interior of a ball mill, 1 metre (40 inches)

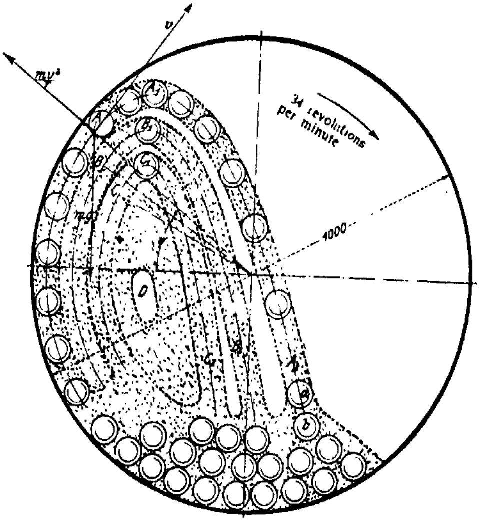

Fig. 176. Diagram of action of tube mill.

in internal diameter, making 34 revolutions per minute. A ball lying against the wall of the drum will be lifted with it until it reaches a position A, where its angular distance above the horizontal is ψ, such that the centrifugal force $\frac{mv^2}{r}$ is less than the component of gravitation, $mg \sin \psi$, which passes from the centre of the drum through the centre of the ball, where m is the mass of the ball, v the linear velocity of rotation in feet per second, r the radius of the drum in feet, and g the acceleration of gravity. At this point the ball will leave the wall of the drum and fall in the parabolic curve A^1A^2, thus striking against the other balls, as at *b*, and crushing the mineral by impact. The same action causes the crushed material to be lifted and gradually pushed forward to the discharge end of the mill.

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[1] *Zeitsch. d. Ver. Deutsch. Ingen.* March 26, 1904, p. 437.

Prof. Fischer appears to attribute the crushing action wholly to percussion, but it seems more probable that crushing is largely performed also by attrition between the balls as well as by impact.

From the above theory of the ball mill the maximum limiting velocity of rotation can be calculated:

Let D be the diameter of the mill in feet, n the number of revolutions per minute, g the acceleration of gravity (32·2 ft. per second per second).

Assuming that the inside of the mill is rough, and that the friction due to the centrifugal force of the rotating ball, $\frac{mv^2}{r}$, pressing against the inside is sufficient to keep the ball from slipping, so that it moves round with the same speed as the mill, the maximum possible value for v, at which the ball will not be carried round continuously is given by the expression $v = \sqrt{rg}$. When the ball reaches the highest point of its course, $\psi = 90^\circ$ and $\sin\psi = 1$, or $mg\sin\psi = mg$. Equating the vertical action of gravity against centrifugal force,

$$mg = \frac{mv^2}{r} \text{ or } v = \sqrt{rg}.$$

But

$$v = \frac{n\,2\pi r}{60}.$$

Whence

$$n = \frac{60\sqrt{rg}}{2\pi r} = \frac{54{\cdot}18}{\sqrt{r}},$$

or

$$n = \frac{76{\cdot}6}{\sqrt{D}},$$

or if the diameter d be expressed in inches $n = \frac{265{\cdot}5}{\sqrt{d}}$. Thus it appears that the rule above quoted from Mr Davidsen allows an ample margin for safe working as deduced from theoretical considerations.

Mr Hardinge[1] has designed a tube mill which instead of being cylindrical consists of a double hollow cone, i.e. of two conical shells placed base to base, the one having a much steeper generatrix than the other; the common axis of the cones is horizontal. He claims to get a far greater efficiency with this form of mill than with the cylindrical mill, but the device is only now in the experimental stage.

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[1] *Bull. Amer. Inst. Min. Eng.* 1908, p. 581.

Disintegrators. A considerable number of machines depend for their crushing action partly upon the impact of particles of the mineral against each other and partly upon the impact of portions of the machine against particles of the mineral, the latter being in suspension in the air and not resting upon any solid support; such an action is only possible when very considerable velocities are attained. The crushing takes place mainly by percussion, partly also by attrition. It would be as well to apply the term "Disintegrator," by which some of these machines are generally known, to all machines of this type.

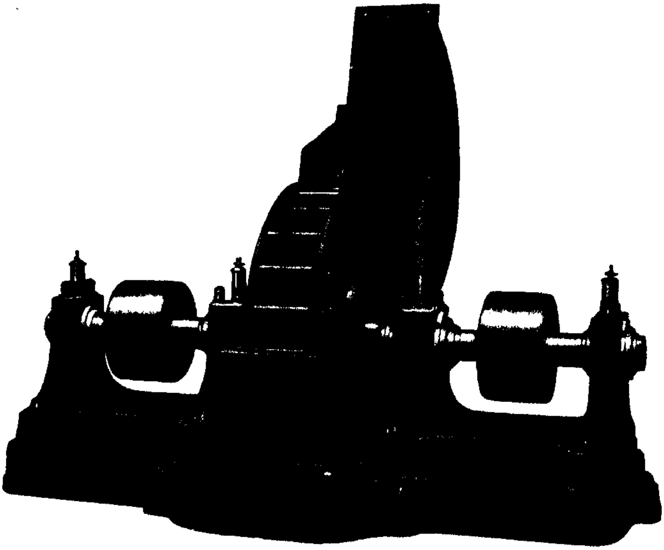

Fig. 177. Disintegrator. Perspective.

The best known of all of them is **Carr's Disintegrator**, known in America as the Stedman and on the Continent also as the Hubner. It consists of two iron discs, each supported by a short length of horizontal shafting carried in suitable bearings and furnished with a driving pulley. Sometimes the two driving pulleys are at opposite ends of the machine, as in Fig. 177, sometimes they are both brought to the same end by making one shaft hollow, as in Fig. 178; the former system is generally preferable. The pulleys are driven one by an open, the other by a crossed belt, so that the two shafts revolve in opposite directions. The two discs are a few inches apart, and on the sides facing each other are provided with concentric rings of iron pins or teeth that reach nearly to the opposite disc, the annular rows of teeth on one disc alternating with those on the other; there are usually three rings of teeth on one, and two on the opposite disc. The discs are enclosed in a substantial iron casing, in the upper part of which is a hopper for the introduction of the mineral to be crushed into the centre of the machine, the discharge being through a shoot in the bottom of the casing. The two discs are caused to revolve at high velocities in opposite directions, the speed of the outer teeth being usually some 7000 feet per minute; a fragment of mineral falling down is struck by a tooth and sent flying off tangentially with considerable velocity, at which it meets the next tooth moving also at great speed in the opposite direction, and is shattered by the impact, this action continuing until the comminuted particles are thrown off from the outer ring of teeth. These machines are not adapted to the crushing of hard material, the wear being excessive and the crushing capacity extremely low; they give excellent results on soft material such as coal, for which they are very largely used. Carr's disintegrators are built in various sizes ranging from 2 feet to 8 feet in external diameter. A 6 foot machine, which is a convenient size, requires about 25 H.P. and crushes from 15 to 30 tons of coal per hour, running at about 200 revolutions

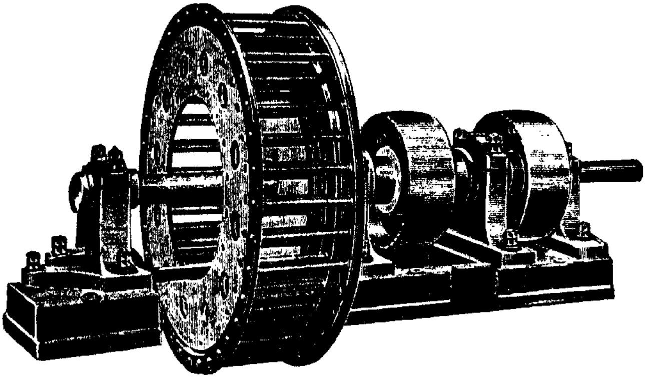

Fig. 178. Disintegrator. Perspective.

per minute. Such a machine weighs about 5 tons and costs about £250.

The Cyclone pulveriser[1] consists of a pair of steel fans revolving in opposite directions in a steel lined drum into which the mineral to be crushed is fed. The fans make 2000 to 3000 revolutions per minute, and at this rate can crush 1 to 2 tons of material per hour with a power consumption of about 20 H.P. This machine is specially adapted to extremely fine grinding.

The Sturtevant mill[2] is very similar in general arrangement; as shewn in Fig. 179 it consists of a horizontal drum lined with screen bars; the two ends are closed by conical bushings which revolve at a high rate; the mineral is fed in at the top and discharged at the bottom of the casing. The mills are made of various sizes, 15 inch and 20 inch in diameter being those most used. The former mill crushes about 16 tons per hour, consuming 70 H.P.; the latter are run at about 900 revolutions per minute, consume from 96 to 115 H.P. and crush 20 to 24 tons per hour to 12 inch mesh. The wear comes chiefly upon the conical bushings which are made of chilled cast iron. A set of these weigh 800 to 1000 lbs. and are variously estimated to outlast the crushing of 600

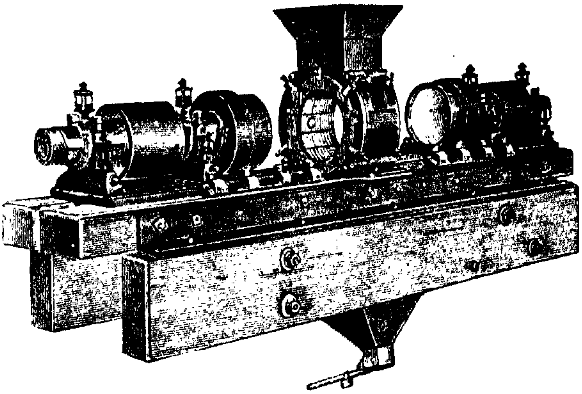

Fig. 179. Sturtevant mill. Perspective.

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[1] *Trans. Amer. Inst. Min. Eng.*, "The Utilization of Puddle- and Reheating-Slags," by Axel Sahlin, Vol. xx. p. 387.

[2] *Trans. Amer. Inst. Min. Eng.*, "Granulating Magnetic Iron Ore with the Sturtevant Mill," by W. A. Hoffman, Vol. xxi. pp. 126, 530, Vol. xx. p. 605.

to 6000 tons of ore. It is stated that the wear is greatly increased when moist ore is crushed, and that the most favourable results were obtained on roasted ore.

Machines working by attrition are employed only where very fine crushing is required; it would probably be best to restrict the term grinding to the designation of this class of crushing, which is but rarely employed in dressing operations, though often used in preparing minerals for chemical or metallurgical operations.

The simplest form of grinding machine is the well-known **Arrastra,**

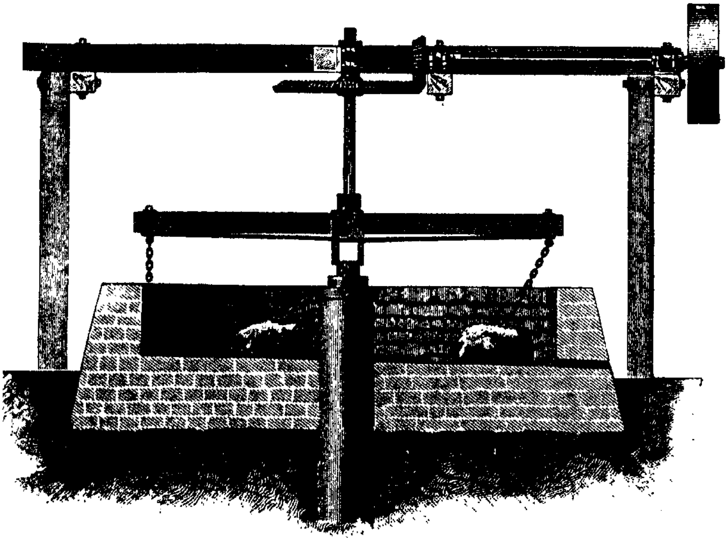

Fig. 180. Arrastra. Vertical section.

which has been largely used for treating gold and silver ores in Mexico and the Western States of America. It consists, in its simplest form, of a shallow pit from 1 to 2 feet deep and 10 to 20 feet in diameter, paved closely with hard stones. In the middle of the pit an upright post is pivoted, to which are attached several (usually four) horizontal arms. From the ends of these arms hang heavy drag-stones, there being usually one to each arm, and the usual weight being 6 to 7 cwt. These stones are attached to the arms by means of chains or raw hide ropes, the line of attachment being somewhat in front of the centre line of the stone. One of the arms projects beyond the edge of the machine and to it a mule is usually harnessed; arrastras are also worked at times by water or steam power. Such an arrastra, made by the Union Iron Works of San Francisco, is shewn in Fig. 180. The arrastra is charged with from 1 to 3 tons of mineral (usually hard quartzose ore), the drag-stones are so adjusted that the front edge is just clear of the bottom, and the machine is set in motion, making from 6 to 12 revolutions per minute. The ore is ground fine in about 2 to 3 hours, water being poured in as soon as it is reduced to about the size of coarse sand.

Improved arrastras have been constructed in which the pan is of iron about 8 feet in diameter, lined with chilled cast iron bottom plates upon which the drag-stones bear.

The **Chilian mill** consisted originally of a bed of stone very much like that of the arrastra, though usually rather smaller in diameter. The central upright carries only one transverse arm, on one end of which revolves a circular stone like a mill stone, mostly of granite, 5 to 6 feet in diameter and about 18 inches wide, whilst a mule is harnessed to the other end. More modern forms are used in which the pan alone, or both pan and roller, are made of iron instead of stone. Such mills are used for crushing soft gold ores in the Ural mountains[1]; they consist of iron pans about 15 feet in diameter and 2 feet 6 inches deep, lined with steel plates, the front portion of the pan carrying screens. The central shaft, driven by gearing, carries three arms at angles of 120° to each other, on each of which is a roller about 4 feet in diameter and 1 foot wide, with steel tyres, weighing altogether about 3½ tons. The shaft makes about 12 revolutions per minute and the power consumption of the mill is about 7 H.P. It crushes about 18 tons per 24 hours of the rather soft ore to one-eighth inch mesh.

In spite of the existence of these improved forms, it may fairly be said that the main advantage of the Chilian mill and the arrastra lies in the fact that the simpler patterns form crushing machines that are capable of being worked by animal power, and are hence useful when no mechanical power is obtainable.

The Bryan roller mill and the Bradley Chilian mill are practically identical in construction with the last-named, and differ only in details; these are made from 4 to 6 feet in diameter. The Union roller mill and the Griffin ore mill are similar, but have rollers that incline inwards so as to counteract centrifugal action, the grinding surfaces of the rollers being not cylindrical, but very short frustra of cones.

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. XXVIII. 1898, pp. 31, 846.

The common mortar mill, which is exceptionally used for crushing minerals, differs from the Chilian mill in that the pan is caused to

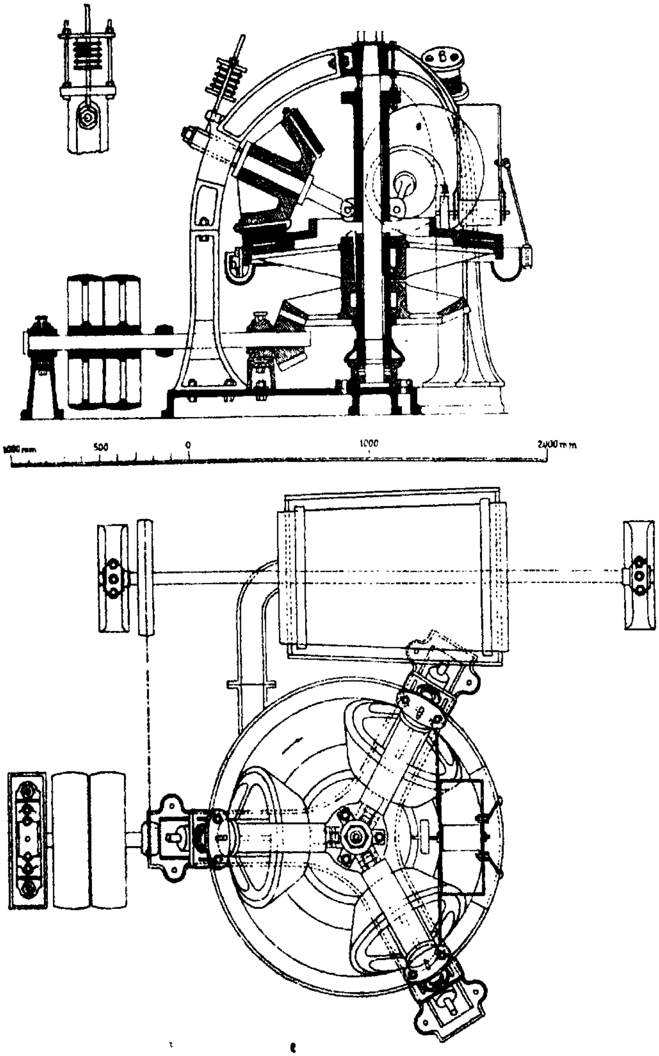

Fig. 181. Schranz mill. Vertical section and plan.

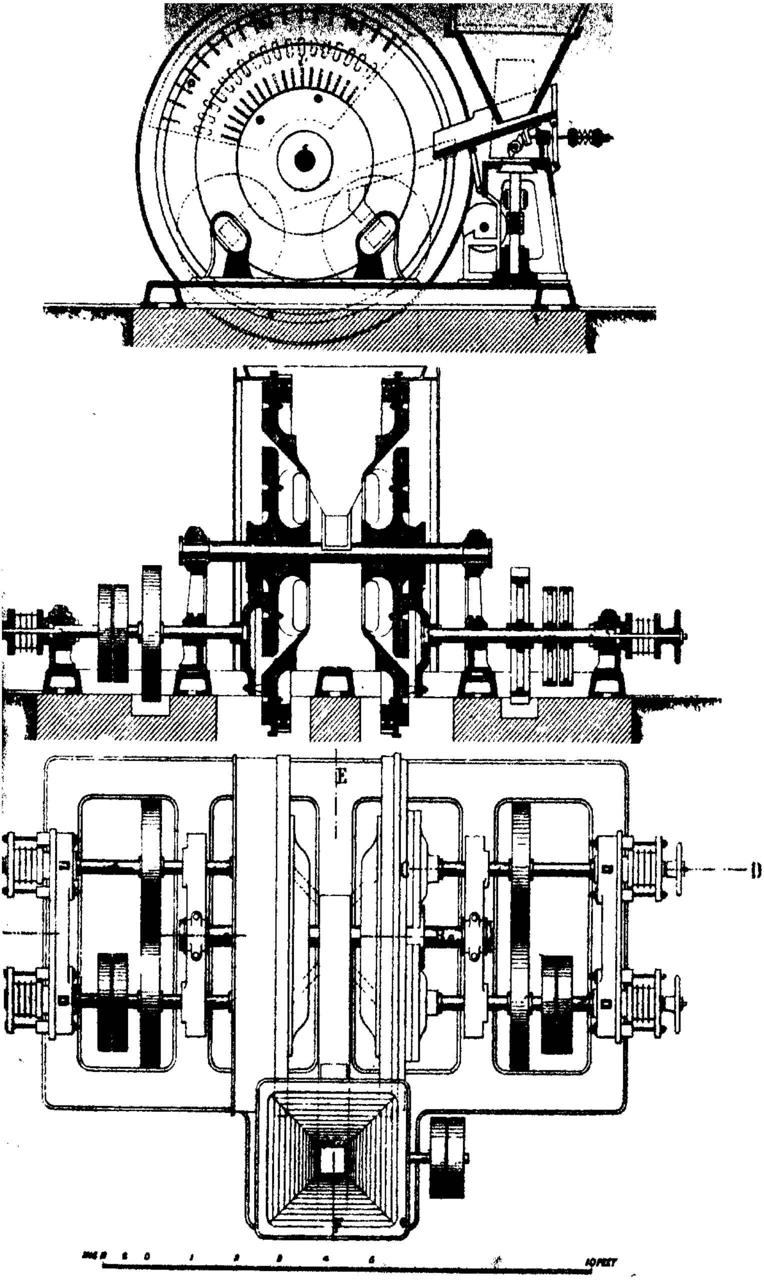

Fig. 182. Heberle mill. Vertical section, plan and side elevation.

revolve by suitable machinery, the rollers which rest upon the pan revolving owing to the friction between them and the pan.

The **Schranz roller mill**, Fig. 181, depends also upon the last mentioned principle. It consists of a disc about 4 feet 6 inches in diameter, the upper surface of which forms a flat cone, the generatrix of which is inclined about 10° to the horizontal; this disc revolves at 12 to 14 turns per minute. Upon its upper surface rest three rolls having the form of frustra of cones, the generatrices of which form an angle of about 10° with their axes. These conical rollers are free to revolve, their axes being pressed down by means of powerful springs by which any desired degree of pressure can be produced. The diameter of the rollers is about 2 feet 3 inches, and they make from 30 to 36 revolutions per minute; they are provided with renewable steel mantles. These mills require 3 to 4 H.P.; they are chiefly used for the re-crushing

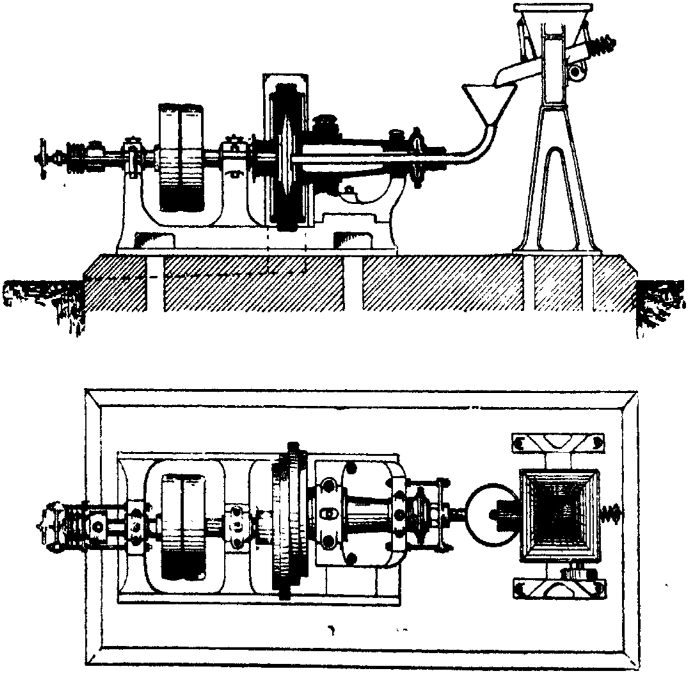

Fig. 183. Simple Heberle mill. Vertical section and plan.

of middlings, and have been found capable of crushing 8 tons of hard quartzose ore per 24 hours from 0·3 inch down to 0·08 inch mesh.

The **Heberle mill**, shewn in Fig. 182, consists of a vertical disc about 6 feet in diameter rotating at about 2 revolutions per minute, against which press from 1 to 4 smaller discs 28 inches in diameter, making 250 revolutions per minute. The surfaces of these discs, where they are in contact, are often grooved so as to increase their grinding power. These mills consume about 5·25 H.P. and crush about 17 tons for each disc per 24 hours; they are used exclusively for re-crushing ore that is already broken to less than 0·5 inch. The figure shews a double mill, consisting of a pair of large discs keyed on one shaft.

A simpler form, consisting of one revolving disc grinding against a fixed disc, is shewn in Fig. 183, as made by the Humboldt Engineering Co. This mill is said to be capable of grinding 6 to 8 cwt. of ¾ inch stuff per hour down to less than $\frac{1}{10}$ inch; it requires from 5 to 7 H.P. and about 4½ gallons of water per minute.

Pans of various forms, of which the Wheeler pan is one of the best known, are used for fine grinding crushed gold and silver ores, the object being however rather the chemical operation of amalgamation than the mechanical action of comminution. They consist of cylindrical pans with an iron bottom on which revolves an iron muller. They are fully described in the author's *Handbook of Gold Milling*, already referred to.

CHAPTER V.

SEPARATION BY SPECIFIC GRAVITY.

It was pointed out in the opening chapter that a large majority of the methods employed in the separation of minerals from each other depend upon differences in the respective specific gravities of the minerals in question. This operation is performed by the aid of fluids, either liquid or gaseous; in the former case the fluid may have a higher specific gravity than one of the ingredients to be separated, or it may be of lower specific gravity than any of them; in the latter case the specific gravity of the fluid is necessarily always much lower than that of any mineral constituent.

We will confine our attention in the theoretical discussion of the subject to the separation of only two mineral substances from each other, because in practice such methods of separation nearly always take the form of separating out one mineral ingredient (the heaviest or the lightest as the case may be) from a mixture of a number of minerals, then separating out another member of the remaining group, and so on, so that even a complex case of separation is generally resolved into a succession of binary separations.

The theoretically simplest case is where a liquid of intermediate specific gravity is employed. Thus in a mixture of coal (of sp. gr. 1·3) and shale (of sp. gr. 2·5), the former could be separated completely from the latter by crushing the mineral to a degree of fineness such that each particle shall consist wholly either of coal or of shale, and then throwing the crushed mass into a saturated solution of magnesium chloride of sp. gr. 1·33, in which the coal would float, whilst the shale would sink. No mechanical arrangement would be required except some simple form of stirrer to prevent particles of shale from being mechanically entangled and buoyed up by particles of coal, or, *vice versa*, particles of coal from being entangled in and carried down by particles of shale. However simple and efficient such a method is in theory, and in spite of its working admirably on a small scale—

e.g. for the separation of a few grains of mineral in the laboratory—it is quite unsuited for practical application on a working scale. In the first place the range of the method is very limited, because there are but few minerals light enough to float in any of the ordinary available solutions. The mineralogist uses a number of heavy solutions (e.g. Sonstadt's solution, a solution of Mercuric Iodide in Potassic Iodide, sp. gr. 3·19; Methylene Iodide saturated with Iodoform, sp. gr. 3·6; Rohrbach's solution, a solution of Mercuric Iodide in Baric Iodide, sp. gr. 3·58; Klein's solution, Boro-tungstate of Cadmium, sp. gr. 3·28) for the laboratory separation of minerals, but these solutions are very costly, difficult of preparation, easily decomposed, or for other reasons practically impossible to work with on any large scale. Even in the case of such minerals (e.g. coal) as are sufficiently light to float in solutions of cheaper substances (Calcic, Magnesic, or Sodic Chloride), the first cost of the material would be considerable, whilst the loss of substance in the operation itself and the necessity for a thorough washing of the mineral recovered would make the process too costly to be practicable. This method is therefore never used, unless we include in it certain methods of gold extraction, in which a pulp of finely divided material, containing free gold, suspended in water, is allowed to stream through a bath of mercury. The gold, more or less completely amalgamated by the mercury, sinks to the bottom of the containing vessel, whilst the barren pulp floats off at the surface.

The remaining methods in which the minerals to be separated are treated in fluids—liquid or gaseous—lighter than themselves constitute the bulk of the processes with which we have to deal. It is obvious that in these methods separation cannot take place in a state of rest, and either the mineral substances, or the enveloping fluid, or both, must be in motion.

(1) The simplest and one of the most common cases is where the minerals are allowed to fall through the fluid under the action of gravity. It is known that when bodies fall in a vacuum under the action of gravity they fall with uniformly accelerated velocity, the equations connecting the velocity of falling (v), the time of falling (t), and the distance fallen through (s) being:

$$(1)\quad v = gt,$$

$$(2)\quad v^2 = 2gs,$$

$$(3)\quad s = \tfrac{1}{2}gt^2,$$

where g is the acceleration due to gravity (32·2 feet or 981 centimeters per second per second). When a body falls in vacuo, its rate of falling is independent of its weight or volume, because whilst the accelerative f of gravity increases with its mass, the quantity of matter to be m increases in the same proportion, and there being no resistance t overcome, it is unimportant how great its surface may be. The is very different when a body falls through a resisting medium; force producing acceleration depends as before on the mass of substance, but this force has now to perform not only the wor moving the mass but also of overcoming the resistance of the med in which it moves; the latter may be looked upon as including work required to be done in displacing as much of the fluid med as is requisite for the passage of the moving body together with tha overcoming the viscosity of the fluid.

Neglecting viscosity for the moment and assuming for the s of simplicity that the body falling in a fluid is a sphere of diamete and of specific gravity S, and that w is the weight of unit volum water, the weight of the sphere will be $\frac{\pi}{6} D^3 S w$ and in a vac this is the weight producing acceleration.

Let the body of diameter D inches and specific gravity S fall fluid of specific gravity s:

Then the weight causing acceleration is the above weight less tha an equal volume of water or

$$\frac{\pi}{6} D^3 w (S - s).$$

The resistance of a fluid to the fall of a body within it varies in part the square of the velocity with which the body is moving, and in p directly with the velocity. When the velocity is considerable, resistance varying with the square of the velocity need alone be c sidered, when on the contrary it is very small, only that varying dire with the velocity is of much importance, and this only comes into p at velocities so low that the viscosity of the fluid affects the result. considerable velocities the resistance is given by practically all writ on the subject in the following form

$$K A w s \frac{V^2}{2g},$$

where A is the projected area of the body in a plane perpendicular the direction of motion, and V the velocity of motion per second; a coefficient, which is independent of the units employed. Fo sphere, according to Rankine[1] $K = 0{\cdot}51$; Cotterill gives $K = 0$

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[1] *Applied Mechanics*, p. 598.

H. S. Allen[1] gives $K=0{\cdot}54$; Unwin gives values varying from $0{\cdot}31\times 1{\cdot}43=0{\cdot}44$ to $0{\cdot}31\times 1{\cdot}86=0{\cdot}58$. Its exact value does not greatly affect the present question, so that it will be near enough if, following several other writers, the value $K=0{\cdot}5$ be here adopted. The resistance of the fluid, using the above units is therefore

$$0{\cdot}5\left(\frac{\pi D^2}{4}\right) ws\frac{V^2}{2g},$$

where V is the velocity of motion of the sphere in inches per second and g is $12\times 32{\cdot}2=386$ inches per second per second. When the weight causing acceleration is equal to the resistance of the fluid, the velocity becomes uniform; hence equating:

$$0{\cdot}5\left(\frac{\pi D^2}{4}\right) ws\frac{V^2}{2g}=\frac{\pi}{6}D^3w(S-s),$$

$$\frac{V^2}{1030}=D\left(\frac{S-s}{s}\right),$$

$$V=32{\cdot}1\sqrt{D\left(\frac{S-s}{s}\right)}.$$

If the velocity is required in feet per second, this becomes

$$2{\cdot}67\sqrt{D\left(\frac{S-s}{s}\right)}.$$

If a cube of side L inches be allowed to fall, the formula becomes modified as follows:

Weight causing motion $=L^3w(S-s)$,

Resistance $=1{\cdot}28L^2ws\dfrac{V^2}{2g}$.

Equating and solving

$$V=24{\cdot}5\sqrt{L\left(\frac{S-s}{s}\right)}.$$

If the velocity is required in feet per second, this expression becomes

$$2{\cdot}04\sqrt{L\left(\frac{S-s}{s}\right)}.$$

When the diameter and velocity are expressed, as is often the case, in meters, the unit of time being still one second, these equations read as follows, D_m and V_m being the respective magnitudes expressed in meters:

$$\text{Weight causing acceleration}=\frac{\pi}{6}D_m{}^3(S-s)w,$$

$$\text{Resistance}=0{\cdot}5\left(\frac{\pi D_m{}^2}{4}\right)sw\frac{V_m{}^2}{2g}=V_m{}^2\frac{\pi D_m{}^2 s}{16\times 9{\cdot}81}w,$$

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[1] *Phil. Mag.*, "On the motion of a sphere in a viscous fluid," by H. S. Allen, 1900, p. 324.

$$V_m{}^2 = 26{\cdot}16 D_m \left(\frac{S-s}{s}\right),$$

$$V_m = 5{\cdot}11 \sqrt{D_m \left(\frac{S-s}{s}\right)},$$

which is the same expression as has been arrived at by a somewhat different method by Bergrath von Rittinger.

For a cube[1] of side L_m (expressed in meters) we have

$$\text{Weight causing motion} = L_m{}^3 (S-s)\, w,$$

$$\text{Resistance} = 1{\cdot}28 L_m{}^2 sw \frac{V_m{}^2}{2g} = \frac{1{\cdot}28 L_m{}^2 s V_m{}^2}{19{\cdot}62} w,$$

$$V_m{}^2 = 15{\cdot}3 L_m \left(\frac{S-s}{s}\right),$$

$$V_m = 3{\cdot}91 \sqrt{L_m \left(\frac{S-s}{s}\right)}.$$

For the velocity of fall of irregular bodies, Rittinger adopts the device of assuming a sphere of diameter such that the volume of the sphere shall be equal to that of the irregular body. Let δ_m be the diameter of such a sphere. Then according to Rittinger the ultimate velocity of fall in meters may be expressed by:

$$3{\cdot}2 \sqrt{\delta_m \left(\frac{S-s}{s}\right)} \text{ for rounded bodies,}$$

$$2{\cdot}25 \sqrt{\delta_m \left(\frac{S-s}{s}\right)} \text{ for flattened bodies,}$$

$$2{\cdot}65 \sqrt{\delta_m \left(\frac{S-s}{s}\right)} \text{ for elongated bodies,}$$

$$2{\cdot}85 \sqrt{\delta_m \left(\frac{S-s}{s}\right)} \text{ on the average.}$$

Furthermore he points out that the ratio of the volume of a particle to the diameters of a perforation (in a sieve) through which it can just pass, is approximately constant for each type of form, hence an approximate ratio can be obtained between the diameter of such a perforation and the diameter of a sphere of volume equal to the volume of a particle that will just pass through the perforation. If the diameter of the perforation be Δ_m, the terminal velocities in meters per second of the falling bodies will become:

$$2{\cdot}73 \sqrt{\Delta_m \left(\frac{S-s}{s}\right)} \text{ for rounded bodies,}$$

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[1] Maurice (*Compt. Rend. Soc. Ind. Min.* 1896, p. 113) gives for rough cubes of minerals falling in water $V_m = 2{\cdot}652\sqrt{L_m(S-1)}$, using the above notation, as the result of some experiments, but his formulas do not agree in other respects with those here given.

$$1{\cdot}92\sqrt{\Delta_m\left(\frac{S-s}{s}\right)} \text{ for flattened bodies,}$$

$$2{\cdot}37\sqrt{\Delta_m\left(\frac{S-s}{s}\right)} \text{ for elongated bodies,}$$

$$2{\cdot}44\sqrt{\Delta_m\left(\frac{S-s}{s}\right)} \text{ on the average.}$$

The writer is inclined to suspect that the average figures thus obtained are upon the whole rather too low, the tendency of modern crushing practice being to devise methods of crushing that will yield rounded granules as far as possible, and that a higher coefficient than 2·44, say at least 2·6, may safely be adopted.

These expressions shew that for a body of given size and specific gravity there is a limiting velocity which can never be exceeded when it is falling in a fluid medium of given specific gravity; theoretically this limiting velocity would only be attained after an infinite time, but a velocity that is practically constant is attained in a very brief period. Mr H. S. Allen has given the following rough rule as approximately correct for a falling sphere:

The terminal velocity is attained in practice when the sphere has fallen through a distance equal to five times the distance required to set up the same velocity in vacuo.

Therefore the constant velocity V will be practically attained after the sphere has fallen through a distance in feet equal to

$$\frac{5\,V^2}{2g}=\frac{V^2}{13} \text{ nearly.}$$

The laws governing the decrease of acceleration during the first portion of the fall, in the period before practically constant velocity has been attained, are almost unknown. It can only be said that at the commencement of the fall, when the velocity is practically zero, the resistance due to the velocity of falling may be disregarded, and the velocity of falling approximates more nearly to that of a body falling in vacuo; hence at the commencement of the fall, the rate of motion is nearly independent of D and approximately varies directly as $(S-s)$.

It follows from the above formulas that two bodies of different diameters D and D_1, and of different specific gravities S and S_1, will fall with equal terminal velocities in a medium of specific gravity s, when their diameters and specific gravities are to each other such that

$$\frac{D}{D_1} = \frac{S_1 - s}{S - s}.$$

Such particles, having equal terminal velocities, are called "equal-falling particles." Strictly speaking, if two equal-falling particles, of different densities and diameters, be released simultaneously at the surface of a deep vessel containing a fluid such as water, they will not reach the bottom of the vessel at precisely the same moment, because the denser of the two will have a greater initial velocity, and will reach its limiting velocity before the lighter particle will do so. If the depth of water is at all considerable, the difference will however bear so small a proportion to the total time occupied in falling, that it may be neglected for practical purposes. In fact in practice it is always assumed that the particles which reach the bottom of a vessel, as above, at the same moment, are equal-falling particles, and whilst not strictly true, this assumption is near enough to the truth to involve no sensible error, if it be borne in mind that of the two particles that reach the bottom of such a vessel simultaneously, the heavier will be slightly smaller than is indicated by the law of equal-falling particles strictly interpreted. It is obvious that if particles of two different minerals of notably different specific gravities and of different sizes be sized by screening through a series of sieves, the perforations of which are so arranged that all the particles passing through one mesh, and not passing the next smaller, shall have diameters such that the ratio of the smallest to the largest shall not be less than indicated by the condition

$$\frac{D}{D_1} = \frac{S_1 - s}{S - s},$$

the smallest particle of the heavier mineral in that group must necessarily attain an ultimate falling velocity greater than that of the largest particle of the lighter mineral; accordingly it would be possible to completely separate all the particles of the lighter from all the particles of the heavier mineral by their rate of falling in water. The ratio of the size of the perforations of the screen to the next larger or smaller, is known as the sieve scale, or the sieve factor, and it is obvious that this ratio should be determined by the expression $\frac{S_1 - s}{S - s}$. For all practical purposes, this may be written $\frac{S_1 - 1}{S - 1}$ for bodies falling in water, and $\frac{S_1}{S}$ for bodies falling in air. It is hence evident that a larger sieve factor can be used to separate bodies falling in water than when they fall in air, and that the latter method is accordingly inapplicable to the separation of bodies, the specific gravities of which are not widely different, unless very close sizing be employed.

Furthermore, it is evident that when particles are separated not strictly by their ultimate velocities, but by the time they take to fall through a deep body of water, the sieve factor may be allowed to be a little larger than that given above, or in other words, separation by specific gravity takes place under conditions rather more favourable than indicated by the formula of equal-falling.

The formulas hitherto considered apply only to particles falling freely in a fluid medium, that is to say, to individual particles falling in a mass of fluid such that the cross-section of the particles is so small a fraction of the cross-section of the fluid that the reaction of the displaced fluid upon the falling particle may safely be neglected. This is never the case in practice, as the number of particles allowed to fall simultaneously is always so great as to occupy a considerable proportion of the cross-section of the fluid in which they fall. This is usually spoken of as the condition of "hindered falling," the exact laws of which are still unknown. It is obvious that a number of particles falling through a fluid of limited area, will produce upward currents of fluid, corresponding to the displacement of the fluid by the solid bodies, and it will be shewn presently that an upward current of fluid offers a definite resistance to the particles affected by it; as in an assemblage, even of equal-falling particles, the denser fall more rapidly at the beginning, it is clear that these are the less affected by such upward current, the hindering effect of which makes itself felt most by the lighter particles. Again the particles interfere with each other, and such interference affects more especially the larger and lighter particles, the smaller particles being able to slip more easily through the falling mass, and the heavier having more momentum to enable them to thrust the others aside. That such hindrance to free falling does actually take place to a very marked extent has been shewn by several investigators. H. S. Monroe[1] has shewn that a sphere falls more slowly in a tube, the diameter of which approximates to that of the sphere, than when it falls in a practically unlimited body of water, and the same result has been obtained more recently by R. Ladenburg[1]. Monroe also found that a number of spheres falling *en masse* in a tube fall with only one-sixth of the velocity of free-falling spheres. The facts are so far clear, but there are not enough data available to enable calculations or formulas to be based upon them. It may however be taken as proved that all the effects of hindered falling are to favour the smaller and heavier as against the larger and lighter particles, and to enable separation to be performed within limits considerably wider than those indicated by the theory of equal falling above given; in practice it is usually possible to work with a sieve factor double of that indicated by the strict equal falling theory, and yet to get perfectly satisfactory results.

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[1] *Trans. Amer. Inst. Min. Eng.*, "English versus Continental System of Jigging," by H. S. Monroe, Vol. XVII. 1888, p. 637.

It has already been stated that all the above formulas are only applicable to bodies above a certain size and whose ultimate falling velocities exceed certain limits. Minerals so finely divided as not to settle with reasonable rapidity in water are spoken of as "slimes." No definition has ever been set up as to the limit below which crushed material should be regarded as slimes; it is generally speaking considerably below 0·25 millimeter (say 0·01 inch). It is perhaps more scientific to accept as the definition of slimes "particles so small that they no longer obey the law of falling through fluids with a terminal velocity varying as the square root of the diameter," or in other words "particles so small that the resistance to their motion is markedly affected by the viscosity of the fluid in which they fall." For these small particles the terminal velocity has been investigated by various writers (e.g. Stokes, *Cambridge Philosophical Transactions*, Vol. VIII.; H. S Allen, "On the motion of a sphere in a viscous fluid," *Phil. Mag.* 1900, p. 324; O. G. Jones, *Phil. Mag.* 1894, p. 451; Lamb's *Hydrodynamics*, p. 533) who all accept the expression, originally due to Stokes[2],

$$V=\tfrac{2}{9}ga^2\frac{\sigma-\rho}{\mu},$$

where a is the radius of the sphere, V the velocity of fall per second, both in centimeters, σ and ρ the densities of the solid and fluid respectively, and μ the coefficient of viscosity of the fluid. (For a body falling in water at 15°C., $\mu=0·0115$ in C.G.S. units.) In this expression the slip between the surface of the sphere and that of the enveloping sphere of fluid is neglected, the formula of Stokes being based on the assumption that no such slipping takes place; moreover the velocity must be so small that all terms involving squares may be also neglected. According to H. S. Allen[1] the critical radius of a particle above which this law no longer holds good is given by the expression

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[1] *Wiedemanns Annalen der Physik*, "Ueber die innere Reibung zäher Flüssigkeiten," by R. Ladenburg, Ser. 41, Vol. XXII. 1907, p. 287.

[2] *Mathematical and Physical Papers*, by Sir George Gabriel Stokes, Vol. III. 1901, p. 59.

$$a = \sqrt[3]{\frac{9\mu^2}{2g\rho(\sigma - \rho)}},$$

so that for a particle of sand falling through water, taking $\sigma = 2\rho$, the critical radius is 0·085 millimeter. He further infers that in ordinary cases of small particles falling in water, if the velocity exceeds 50 to 100 centimeters per second, viscosity may be neglected, and the law of squares may be considered to hold good. If this inference be accepted, the dimensions of particles forming slimes, as above defined, can readily be calculated.

The investigations of the exact laws governing the motions of these small particles is one of excessive difficulty; here it will be enough to note that they no longer obey the ordinary laws of falling given previously and that their rate of settlement is excessively slow for fine particles; indeed a stage can be reached at which the forces tending to cause falling are insufficient to overcome the resistance to motion, hence such particles tend to remain suspended in still fluid, though if carried down to the bottom of the containing vessel there is no force at all tending to make them rise and considerable resistance to their rising, so that once settled they tend to stay steadily at the bottom of the fluid.

(2) When a body instead of falling in a fluid at rest is submitted to an upward current of water, the nature of the action is similar to what it is when the body is allowed to fall, although according to Rankine the force exerted by a fluid current pressing against a solid body is somewhat greater than the resistance experienced by the same body in falling through the same fluid at rest. According to Rittinger the force has been found to be equal to the resistance, and other authors seem to have assumed without question that this is the case. In the absence of conclusive experimental data which are required to settle this matter, the latter assumption will also be adopted here. On this assumption a body will just remain in equilibrium in an ascending water current when the velocity of the current is equal to the ultimate velocity which the body would attain if allowed to fall freely in the fluid at rest. If the velocity of the current be greater or less than this, the body will ultimately rise or sink with a velocity equal approximately to the difference between the velocity of the current and the ultimate

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[1] *loc. cit.* p. 324.

falling velocity of the body. Just as in the case of a body falling in a fluid at rest, a period of acceleration according to some complex and still unknown law precedes the period of uniform motion. It is obvious that an ascending current will therefore separate assemblages of particles of different specific gravities, provided that their sizes are such that the ratio between the largest particle of the lighter and the smallest particle of the heavier substance falls within the limits of D and D' as given by the equation

$$\frac{D}{D'} = \frac{S_1 - s}{S - s}.$$

It is also clear that the practical execution of the separation is facilitated by the rising current, since the velocity can be so adjusted as to lift all the particles of lower specific gravity whilst allowing all those of the heavier substance to sink.

When particles are submitted to a descending current of fluid they tend to fall through the fluid according to the laws of equal falling but are at the same time carried downwards by the downward pressure of the fluid in accordance with the laws already explained. The action of the fluid reinforces that of gravity and the period of acceleration will be shortened, and during this period the action already referred to is accentuated, in virtue of which the rate of falling from rest at the commencement of the fall depends almost wholly upon the specific gravity of the particle. Hence in a descending current, the denser of two equal-falling particles will at first fall far faster than the less dense.

It follows that if a mass of equal-falling particles be subjected to a series of rapidly alternating upward and downward flowing currents, the equal-falling particles can be sharply separated into groups according to their densities, and that this separation can be best accomplished by a rapid upward, and a slower downward moving current, the duration of the action of either being too short to allow the particles to attain their ultimate velocity, and that this action will be intensified if the particles are so near together as to bring them within the condition of hindered falling.

When particles are allowed to fall through a mass of fluid moving horizontally, their path is the resultant of the various forces acting upon them, that is to say, they move in a parabola tending to become ultimately a straight line. It is obvious that the most rapidly falling particles will reach the bottom of the mass of fluid first and will therefore have been exposed for the least time to the force tending to move them horizontally and will therefore have been carried the least distance horizontally from the point of their introduction, whilst the most slowly falling particles will be carried furthest from that point. A horizontally moving mass of fluid will therefore divide up a group of particles of different sizes and densities into groups of equal-falling particles, the division being indicated now not by time or rate of falling, but by the horizontal space traversed by each group. Such a mass of fluid offers therefore an easy and convenient method of sorting into equal-falling groups, the division being better marked the greater the horizontal velocity of the fluid. If the direction of motion of the fluid be not strictly horizontal but diagonally downwards, the same kind of separation will be produced, provided that the horizontal component of the direction is sufficiently great.

(3) The above statements apply only when the depth of the mass of fluid is so considerable compared to that of the diameter of the particles, that the difference between the velocities of the upper and lower surfaces of the current of fluid may be neglected, and that the entire mass of the fluid may be considered as moving with uniform velocity, and that at the same time the régime of equal-falling velocities may be established. If the diameter of the particle be on the other hand great compared to the depth of the fluid, as is the case when particles are carried along an inclined plane by a thin film of fluid, different considerations prevail.

If the inclination of the plane be α, the weight of the particle pressing upon the plane (cf. p. 216) will be

$$\frac{\pi}{6} D^3 (S - s) w \cos \alpha,$$

and if μ be the coefficient of friction between the particle and the plane, the resistance to its motion down the plane will be

$$\frac{\pi}{6} D^3 (S - s) w \mu \cos \alpha.$$

If F be the mean velocity of the water current consequent upon its flow down the inclined plane, it will exert upon the particle a force equal to $K \frac{\pi}{4} D^2 s w \frac{F^2}{2g}$, where K is a coefficient, that appears, as stated on page 223, to somewhat exceed 0·5, according to Rankine, whilst others take it as 0·5. To enable the particle to move, the force must evidently exceed the resistance, or

$$\frac{F^2}{2g} K \frac{\pi}{4} s w D^2 > \frac{\pi}{6} D^3 (S - s) w \mu \cos \alpha,$$

whence
$$F > \sqrt{\frac{4g}{3K} D \frac{S-s}{s}} \mu \cos \alpha,$$
if the particle is to be moved down the plane.

Assuming that the mean velocity of the current is considerably in excess of the figure thus obtained, let there be placed upon a plane in Fig. 184 two particles, *A* and *B*, of different diameters, but which are equal falling in still water, that is to say, whose diameters and densities satisfy the ratio
$$\frac{D}{D'} = \frac{S'-s}{S-s}.$$

The depth of the fluid may be conceived as made up of a series of thin films $c_1, c_2, c_3 \ldots\ldots c_6$, each of which is moving with approximately uniform velocity, throughout its depth, whilst each is moving more rapidly than the film below it, so that the velocity of c_6 will be almost nil, whilst c_1 will be moving with a velocity considerably greater than the mean velocity of the entire sheet of fluid. Obviously the smaller and denser particle *A* will be exposed to slower moving currents than the larger and less dense particle *B*, and will therefore be moved down the inclined plane more slowly than the latter. By suitably adjusting the angle of the plane, and the depth and velocity of the stream of water, it is possible to cause the larger particle *B* to be carried down the plane whilst the force of the currents to which *A* is exposed shall not be sufficient to move it, being less than the value given in the above formula, so that a very complete separation can thus be brought about.

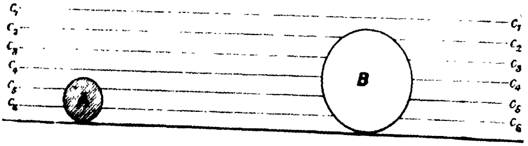

Fig. 184. Diagram of action of shallow currents on particles.

(4) Finally, if a group of particles immersed in a fluid receive simultaneously an external impulse, their momentum will depend upon their mass, and the resistance of the fluid to their motion upon the area of their cross-section; the general case may be stated as follows: let a spherical body rest upon a plane, entirely submerged in a fluid at rest; let this plane receive *n* impulses per second, each imparting the velocity *I* of the plane to the body. If the movement of the plane be now arrested, each impulse will move the body forward relatively to the plane until it is brought to rest by the resistance of the water and the friction upon the plane, the latter remaining at rest.

Let U be the mean velocity of motion of the particle over the plane in the direction of the impulses, and let v be the mean velocity of the particle during the time, t seconds, in which its velocity is reduced from I to zero. All the velocities are measured in units of length per second of time. The problem is not capable of complete solution for want of experimental data, and a number of assumptions must be made, the most important of which is that v will be taken as equal to $\frac{I}{2}$, merely because this value is the mean between its initial and its final values. The remaining notation is the same as on p. 216.

Then the initial momentum of the particle is

$$\frac{\pi}{6} D^3 S w \frac{I}{g}.$$

This momentum is destroyed in time t; the distance traversed by the particle per impulse is vt and per second nvt; whence

$$U = nvt \text{ and } t = \frac{U}{nv}.$$

Hence the force required to destroy the momentum is

$$\frac{\pi}{6} D^3 S w \frac{nvI}{Ug}.$$

And this force is equal to the resistance of the water plus the friction on the plane, these being as before

$$\frac{\pi}{8} D^2 s w \frac{v^2}{2g} \text{ and } \frac{\pi}{6} D^3 (S - s) w\mu.$$

Equating and substituting,

$$\frac{\pi}{6} D^3 S w n \frac{I^2}{2gU} = \frac{\pi}{8} D^2 s w \frac{I^2}{8g} + \frac{\pi}{6} D^3 (S - s) w\mu,$$

whence
$$U = \frac{DSnI^2}{\frac{3}{16} sI^2 + 2gD(S - s)\mu}.$$

From which it appears that when other conditions remain the same a larger particle will be moved more rapidly over the plane than a smaller particle. For particles of the same size but different specific gravities, U increases with S, or the heavier particle moves the faster when $\frac{3}{16}I^2 > 2g\mu D$, and the lighter particle moves the faster when $\frac{3}{16}I^2 < 2g\mu D$. The former is the general condition, so that under most circumstances the heavier particle moves the faster, but in the case of large rough flat particles, where D and μ are both considerable the lighter particle may move the faster over the plane.

In practice the simple arrangement here postulated is never used, but planes subjected to impulses, such as here described, have a current of water flowing over them in a thin film, so that the action described in the last section is compounded with that due to the impulses of the plane, as will be more fully described in Chapter VIII.

A number of processes and appliances are founded upon the above principles; it has been shewn that these are only applicable to groups of particles the dimensions of which exceed certain definite limits, the smaller particles constituting slimes, which have to be treated by another set of appliances, though the line of demarcation between the two groups is by no means sharply drawn. It is evident that by the application of the simple principle of equal falling velocities, all the denser particles could be separated from all the less dense particles of a group in which the ratio of the largest to the smallest diameter shall be less than the ratio $\frac{S' - s}{S - s}$, because in this the smallest particle of the denser material must still fall more rapidly than the largest particle of the less dense. This is usually attained by crushing the crude mineral in any suitable machine, and by passing the crushed product through a series of any of the sizing appliances already considered. The Coxe gyrating screen and others of that class have been used, as also have screens of the Ferraris type, but trommels (p. 55) are to-day almost universally employed for this purpose. It is obvious that the diameter of the screen apertures or meshes must form a series in geometrical progression, the factor of which is the above ratio, which then constitutes the sieve scale or sieve factor. Rittinger recommends a sieve factor of $\sqrt{2} = 1{\cdot}41$, which is thoroughly efficient for most purposes, but is liable to the serious objection of requiring a very large number of trommels and a correspondingly large number of appliances for treating the products of each. His sieve factor is unnecessarily small for most purposes, because as has already been indicated, and as will appear more particularly afterwards, the greater portion of the appliances in use separate particles under conditions of hindered falling, and before the equal falling régime has become established, and can hence separate particles of unequal density when these should not be separable according to the laws of equal falling. A sieve scale of two, or at times even more, may therefore be used with perfect confidence in most cases.

The real difficulties incident to separating lie essentially in the operation of crushing and are referable to the production (*a*) of slimes, and (*b*) of middlings.

(*a*) When a complex mineral mass is crushed small enough to isolate the different individuals of which it is composed, which is the indispensable preliminary to separating, a certain portion is always unavoidably crushed finer than necessary, producing more or less material so finely divided as not to obey the usual laws of settling—which, as already stated, is known as slimes; a good many types of machinery have been devised for treating these slimes, but so far none can be said to have attained complete success; they mostly depend upon the principle of causing the material to settle by agitation in very thin sheets of water. It is hence important to select crushing machinery that yields a granular product and that makes as little very fine powder as possible; whence the need of avoiding machines that act by abrasion if the product is to be treated by these methods of separation.

(*b*) When such a complex mineral is crushed, not only are the individual species isolated from each other, but a certain number of particles will always be produced, consisting partly of the lighter and partly of the heavier mineral, and having therefore a specific gravity intermediate between the two. Such particles will necessarily behave just like particles of some individual mineral of the same specific gravity as the composite particle, and it is impossible to dress such composite particles except by crushing them again so as to liberate the individual constituent parts. As the position occupied in dressing operations by a particle of given size depends only on its specific gravity, nothing is gained by repeated dressing operations applied to the same particle, and Prof. Richard's[1] dictum "once middlings always middlings" necessarily holds good, always provided that these are "true" middlings, due to the presence of both mineral individuals in the same particle; in practice "false" middlings are sometimes met with, due to some imperfection in the dressing operation. These latter can be separated by repeating the same dressing operation, and to these the above statement does not apply

Accordingly in treating a complex mineral, although it contain only two constituents, it is necessary to provide for three classes of products, namely, the mineral of higher density, that of lower density, and the middlings, the latter class having afterwards to be crushed down until it has been split up into its constituents; this crushing is necessarily accompanied by the production of slimes and consequently entails loss. The best practice will therefore be to do as little crushing as possible; this is accomplished by the method of "gradual reduction," which consists in first crushing the mineral to such a size as will liberate the coarsest individual particles present, screening out this coarser size, dressing it, and crushing the middlings finer, the same being done with each successive size of particles. Here however it must be insisted that the operations to be employed in any given case depend essentially upon the commercial values of the products. The theoretically best method of separation is not the one that must necessarily be employed, but rather the method that gives the best economic results. Hence processes that are to be highly recommended for valuable bodies like tin ores or gold ores would be utterly out of place for cheap materials like coal or iron ores. It is this important consideration that renders it impossible to draw up any scientific classification of dressing methods, and that causes the same process to be used in one case as a preliminary, and in another as a finishing—perhaps as the only—process.

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[1] *Amer. Inst. Min. Eng.* Vol. xxii. p. 700.

A rough classification of wet dressing processes is as follows:

Processes that submit minerals to the action of

A. Water currents that are in the main vertical and in one direction:

1. Preliminary: Spitzkasten, Spitzlutte.
2. Finishing: Syphon washer, Robinson washer.

B. Water currents that are in the main vertical and in alternately opposite directions:

Jigs.

C. Water currents that are in the main horizontal:

1. Preliminary: Buddles, tyes, sluices.
2. Finishing: Trough washers.

D. Water currents in thin sheets:

Frames, tables, blanket strakes, etc.

E. Water currents in thin sheets, aided by external impulses:

Shaking tables, Vanners.

Of these groups, *A*, *B*, and *C* are unsuited to the treatment of slimes, which can be treated by appliances of the types *D* and *E*, some of those included under type *D* being slime machines *par excellence*.

CHAPTER VI.

APPLIANCES DEPENDING ESSENTIALLY ON VERTICAL FALL.

THESE appliances may be separated into two groups, the first embracing such as are used for preliminary treatment of crushed mineral, the second such as are used for final dressing.

A. APPLIANCES FOR PRELIMINARY TREATMENT.

This is only applied to particles of mineral so fine as to present difficulties in sizing, and therefore very often to the undersize from the finest mesh of a set of trommels or other sieving appliances; it treats particles that are usually less than $\frac{1}{25}$ inch (1 millimeter), but is exceptionally used for particles up to $\frac{1}{16}$ inch (1·5 millimeters) in diameter. Its object is to divide the mass of mineral particles into groups of practically equal falling particles, each of which groups is to be subsequently submitted to suitable dressing operations. This process of separating into equal falling groups is best spoken of as *classification* (German *Sortirung*), whilst an equal-falling group may be described as a *class*. Generally the pulp, consisting of particles suspended in water, after passing through the finest screen, flows to the classifying appliances, three or four classes being usually made; the escaping water contains then only the finest slimes which are allowed to settle as completely as possible in such simple devices as slime pits, labyrinths, etc.

The appliances used in classifying are the spitzkasten, called also V box or pointed box, the spitzlutte and various modifications, which latter refer entirely to details of construction and not at all to essentials, these latter remaining just as they were when first designed by Bergrath von Rittinger.

Spitzkasten. A spitzkasten in its simplest form consists of an inverted pyramidal box rectangular in plan, and having an aperture at the apex. If a stream of pulp be allowed to flow slowly in at one side of this box as indicated in the diagrammatic section, Fig. 185, at *A*, and flow out again at *B*, it will follow approximately the direction indicated by the arrows and will deposit in the box all the particles capable of falling through the belt of moving water of maximum depth *CD*, during the period that the pulp occupies in flowing horizontally from *A* to *B*. The class of particles deposited in the box will depend therefore upon the depth of the belt of moving water which is partly controlled by the difference in height between the inflow and outflow levels, and partly by the time occupied by the pulp in flowing along the box, which latter element again is controlled by the length of the box and its breadth for a given bulk of pulp, and by the speed of the inflowing current. Rittinger prefers that the inflow and outflow shall be on exactly the same level, and Professor Richards[1] has shewn experimentally that this opinion is sound, and the latter has furthermore shewn that the angle of the inflowing pulp should not exceed 5° to the horizontal. His interesting investigation shews that in all cases eddies are set up and that currents exist in the spitzkasten other than those indicated in the above theoretical diagram, and that these currents carry down material finer than that which should normally settle in the box, so that it is impossible to produce complete classification. The spitzkasten gives however an approximate classification and is still a good deal used on account of its cheapness and simplicity. The conditions under which it does satisfactory work have been determined from purely empirical data. It may be pointed out that although the mineral to be classified is brought in by a horizontal stream of water, the separation takes place under conditions of vertical falling, so that the appliance properly belongs to the group now under consideration.

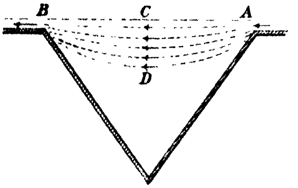

Fig. 185
Diagram of action of Spitzkasten.

It is usual to arrange a series of such pointed boxes, each being so much wider and longer than the previous one as to give a distinctly different class of product. Usually there are three or four such boxes in series. Rittinger's rule is that the first box shall have a width of $\frac{1}{10}$ foot for every cubic foot of pulp flowing into it per minute and a length of about 6 feet; the subsequent boxes have widths increasing in geometrical progression with a ratio of 2, whilst the lengths increase in arithmetical progression by 3 feet at a time. Thus the dimensions of a set of four boxes to treat 20 cubic feet of pulp per minute would be on this principle:

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[1] *Trans. Amer. Inst. Min. Eng.* xxvii. p. 249; *Ore Dressing*, Vol. i. p. 434.

	I.	II.	III.	IV.
Length	6′	9′	12′	15′
Breadth	2′	4′	8′	16′

Linkenbach recommends that for each 10 cubic feet of pulp per minute, the first box should have a width of 25·5 inches and a length of about 12 feet, successive boxes increasing in width in geometrical progression with a ratio of 1·5, and in length in arithmetical progression by 4 feet at a time. A set of four boxes to treat 20 cubic feet of pulp per minute would therefore have the following dimensions:

	I.	II.	III.	IV.
Length	12′	16′	20′	24′
Breadth	4′ 3″	6′ 6″	9′ 9″	14′ 8″

He points out however that when much water is drawn off with the sands the rate of flow is diminished by this circumstance, and that under these conditions notably smaller dimensions can be employed than those given by his formula.

The sides of the box must have slopes of not less than 50° to the horizontal, in order that none of the sands may be deposited upon them; the lower portion of the box is therefore made in the shape of a square pyramid having the required angle, the shorter sides continuing at this slope, whilst the larger sides are made vertical as soon as they have reached the desired width. A baffle board should be placed across each box, dipping about an inch below the water level to prevent any particles being carried straight across to the overflow. The overflow may be 3 to 6 inches below the intake; both should be accurately horizontal, and the latter is best furnished with "pins" so as to give a uniform supply to all parts of the box. For the largest sizes the boxes would necessarily have to be very deep if the minimum angle of 50° is to be maintained; this objection is often got over by making them double or treble pointed, so that the lower portion consists of two or three equal square pyramids having the desired slope. If the last box would otherwise be of unwieldy dimensions the pulp may be divided equally amongst two (or more) boxes each having only one half (or a proportionate fraction) of the width that would be necessary according to the formula.

The classified sands are drawn off at the apex; if this be provided with a simple aperture, the sands will be diluted with an excessive amount of water, owing to the very considerable hydrostatic head produced by the depth of water in the box, unless the aperture be very small, in which case there is danger of its becoming stopped up too readily. This objection is avoided by either furnishing the outlet with a valve gate which is only opened from time to time when a considerable amount of sand will have accumulated (which may be done automatically, e.g. by the Ayton Intermittent Extractor, manufactured by the Allis Chalmers Co.), or else, and more usually, by means of a syphon tap. This consists of a pipe leading from the apex upwards to any desired height; the rate of flow of the pulp from this pipe is regulated by the depth of the discharge below the level of the water in the box, being less of course in proportion as this depth diminishes. The pipes are supplied with plugs to enable them to be cleaned out readily and any accidental stoppage removed.

The pulp going to the first spitzkasten usually carries from 14 to 21 lbs. of sands to the cubic foot.

Fig. 186 shews the simplest form of spitzkasten on the Rittinger system, as used at an Australian mine, built entirely of wood, a pattern that is often adopted with but little if any modification, as it is easily and cheaply built, can be erected and repaired by any mine carpenter, and if properly looked after will last in good order for 10 years or more.

The spitzkasten shewn in Fig. 186 is capable of treating ½ ton of material per hour, or 7 cubic feet of pulp per minute; No. 1 box is a very small one, intended only to catch specially coarse fragments, that ought not really to find their way into the pulp at all. The spigots are formed of short pieces of 1¼ inch iron pipe to which a piece of india-rubber hose pipe is attached, the end of which can be raised or lowered to regulate the speed of outflow of the classified pulp.

A more modern form designed by Messrs Fraser and Chalmers, Ltd., but shewing the same essential form of construction with adjustable quadrant spigots, is shewn in Fig. 187.

Occasionally these boxes are built of sheet iron and may then be either pyramidal or conical; it must be remembered that when the minerals contain much pyrites, acids may be generated by the oxidation of these substances when finely crushed, and under these conditions iron boxes are apt to be rapidly corroded.

It is now very usual to work spitzkasten with an additional upward current of clear water, sometimes spoken of as "hydraulic" water. This

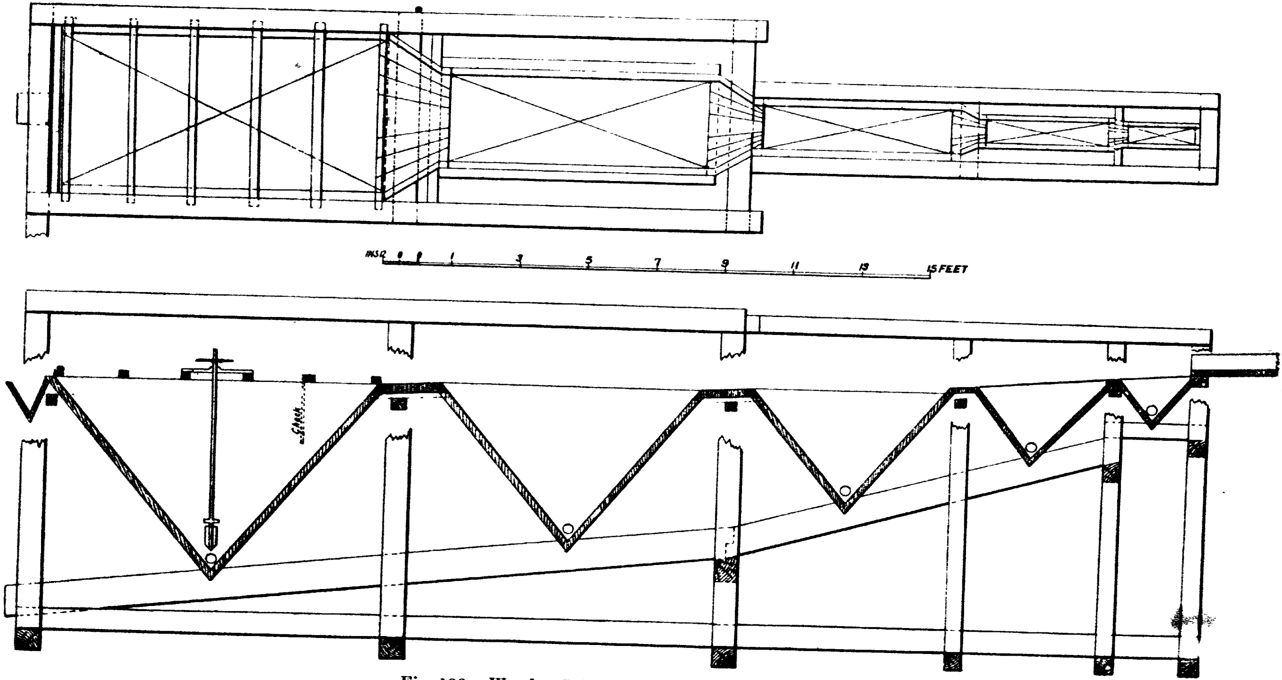

Fig. 186. Wooden Spitzkasten. Plan and vertical section.

is done by bringing a vertical water pipe within 2 or 3 inches of the bottom of each box, either inside or from the outside of the box, all these pipes being connected with a water main, and each having its own stop valve, by which the flow of water can be regulated. By this means (as explained on p. 223) a sharper classification can be produced, the upward current being so adjusted that only the required class of mineral grain can drop past it, whilst it also washes off any very fine slimes that might adhere to the larger grains. These spitzkastens are more often made in the form of a continuously widening trough divided into compartments by transverse partitions, each compartment running out below into a pyramidal form. Jacométry and Lenique's cast iron classifier[1], Fig. 188, is of this type, as also is the

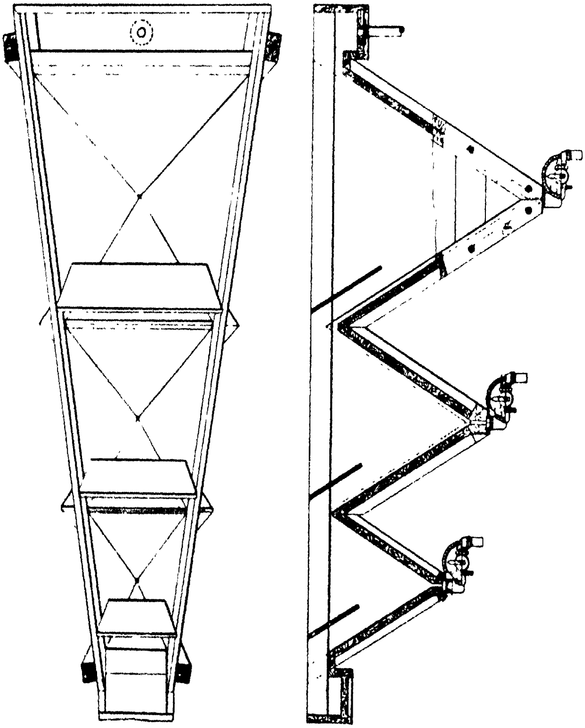

Fig. 187. Spitzkasten with quadrant spigots. Plan and vertical section.

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[1] *Ore and Stone Mining*, C. Le Neve Foster, 4th edn. 1901, p. 587.

Calumet or Richards-Coggin classifier[1], Fig. 189, the latter being used at the Lake Superior Copper mines. The Calumet classifier has an

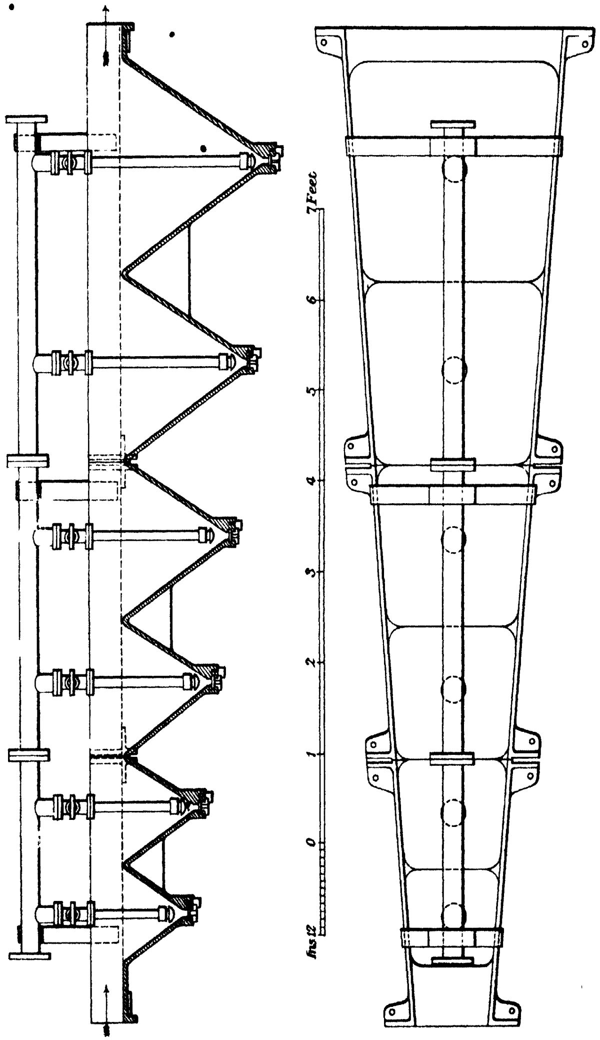

Fig. 188. Cast iron classifier. Plan and vertical section.

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[1] R. H. Richards, *Amer. Inst. Min. Eng.* Vol. xi. 1883, p. 231.

unusually deep baffle board, S, whilst the shield placed just above the nozzle of the clear water inlet is also a notable improvement. This classifier consists of a trough 9 to 18 feet in length, widening gradually from about 8 inches at the feed end to 12 inches at the delivery end, and set at a gradient of about 1 in 8; in the trough are from 3 to 5 (usually 4) wedge shaped depressions usually about 12 inches deep, and 18 inches to 2 feet wide at their upper end; these act like a spitzkasten and each receives an elbow pipe, which supplies it with hydraulic water, the pipes being about 1 inch in diameter. The position of the baffle boards above the ends of these pipes is adjustable. Opposite the ends of the pipes is a simple spigot from which the concentrated products are discharged. According to Professor Richards[1] this classifier, working on $\frac{3}{16}$ inch steam-stamp stuff, will treat 60 to 65 tons in 24 hours with a water consumption of 700 to 800 gallons per ton of ore, and yielding approximately

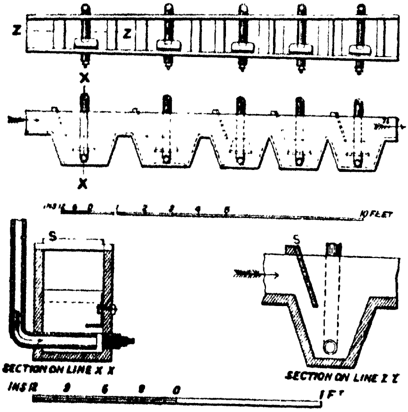

Fig. 189. Calumet classifier. Plan, elevation and sections.

from the first box	20 tons	per 24 hours.
„ „ second „	12 „	
„ „ third „	8 „	
„ „ fourth „	5 „	
overflow	15 „	

There are a large number of very similar classifiers in use in the

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[1] *Ore Dressing*, Vol. I. p. 392.

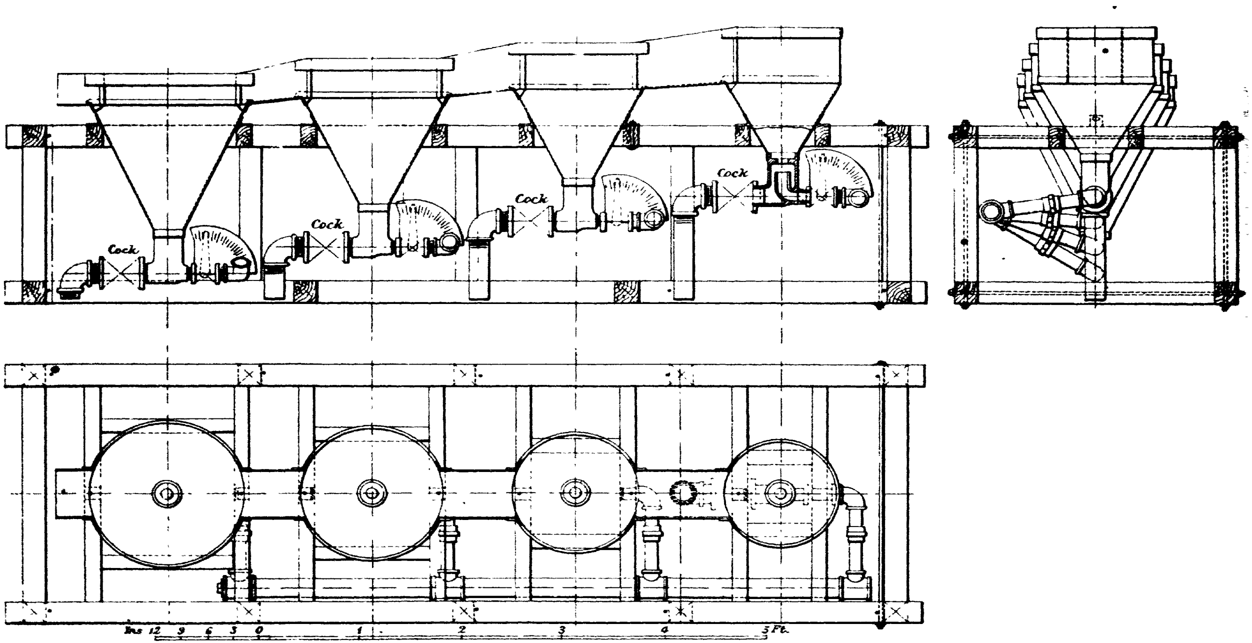

Fig. 190. Browne hydrometric classifier. Plan, side and end elevations.

Lake Superior district. Prof. Richards has brought out an improved form known as Richards' Shallow Pocket classifier, in which the hydraulic water is introduced through a cone which forms the bottom of each of the depressions, the hydraulic water entering the cone tangentially, so as to rise up with a swirling motion and thus effect a better separation of the slower- from the faster-falling material. The capacity of this appliance appears to be about the same as that of the Calumet classifier. The Browne Hydrometric classifier is a spitzkasten with an upward flow of hydraulic water, the separate boxes being cones of earthenware or occasionally of iron. It is shewn in Fig. 190 as manufactured by Messrs Fraser and Chalmers, Ltd., but may be made with either 3 or 4 cones. The manufacturers recommend one of these appliances to every 10 head of stamps, or say to treat 30 tons in 24 hours, the material being crushed to pass about a 30 mesh screen.

Linkenbach recommends as suitable dimensions for a spitzkasten with hydraulic water a breadth of 8 inches for each 10 cubic feet of pulp per minute for the first box, this box to have a length of 20 inches, whilst the lengths and breadths of successive boxes are to be 1½ times the preceding ones. Thus to treat 20 cubic feet of pulp per minute a set of three boxes would have the following dimensions:

	I.	II.	III.
Length......	1′ 8″	2′ 6″	3′ 9″
Breadth ...	1′ 4″	2′	3′

Each box will then require an amount of hydraulic water equal to 0·3 of the volume of the pulp, delivered under a pressure equivalent to a head of about 12 feet.

At Schemnitz, Hungary, 20 tons are passed in 24 hours through a spitzkasten plant consisting of four boxes, using hydraulic water, of the following dimensions:

	I.	II.	III.	IV.
Length......	6′	9′	12′	15′
Breadth ...	2′ 9″	5′	8′	15′
Depth.........	4′	6′	8′	10′

The typical **Spitzlutte** consists of a box in the shape of an inverted triangular prism, within which is set a similar, but smaller box, the position of which is capable of adjustment. A variable space is thus left between the inner wall of the outer box and the outside of the inner box; pulp is admitted at *A* in the diagrammatic section (Fig. 191), passes down one limb of the V-shaped channel left between the two boxes, and overflows at *B*, any material, the ultimate falling velocity of which exceeds the upward velocity of the current in the second limb collecting in the apex of the box, whence it is drawn off as in the case of the spitzkasten. The effective depth of water in which the fall takes place being greater than in the spitzkasten, separation takes place more completely in accordance with the laws of equal-falling particles. The arrangements for drawing off the classified particles from the apex of the box are the same as in the spitzkasten, and, as in the latter, a jet of clear water ("hydraulic" water) may also be introduced at the apex. This appliance will work on thicker pulp than the spitzkasten and is also capable of more thorough adjustment, because in addition to varying the quantity and degree of dilution of the entering pulp and the quantity of clear water as in the spitzkasten, the width of the channel is also capable of regulation. A set of spitzlutten can therefore be all of the same size, the difference in the fineness of the

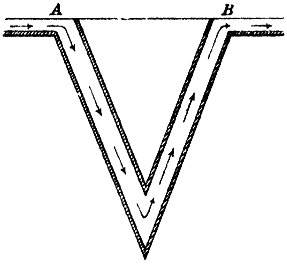

Fig. 191.
Diagram of action of Spitzlutte.

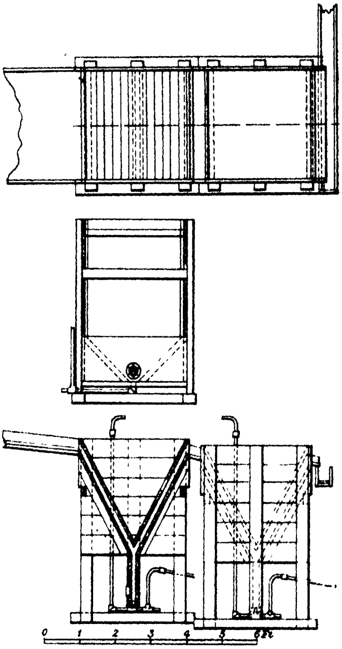

Fig. 192. Wooden Spitzlutte.
Plan, elevation and sections.

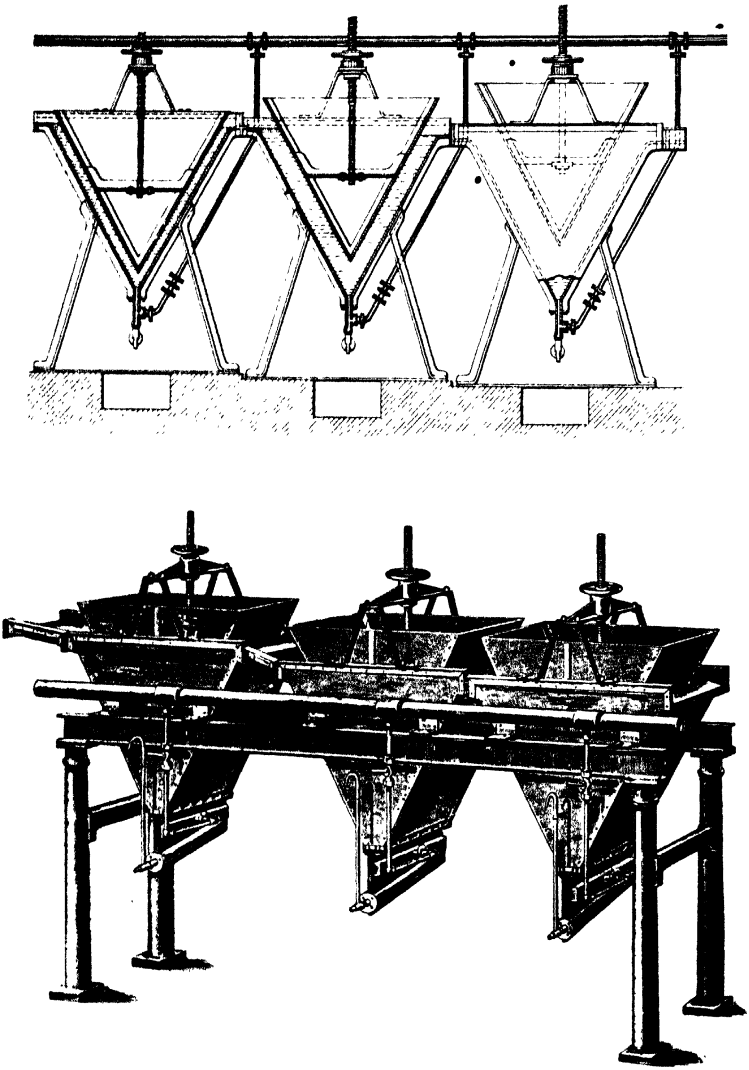

Fig. 193. Iron Spitzlutte. Sectional elevation and perspective.

classified sands being produced by the varying positions of the inner box. The spitzlutte is perhaps more largely employed in modern practice than the spitzkasten; a simple wooden form is shewn in Fig. 192, and an improved form as made by the Grusonwerk Company, is shewn in section and in perspective in Fig. 193, the entire plant being here made of iron.

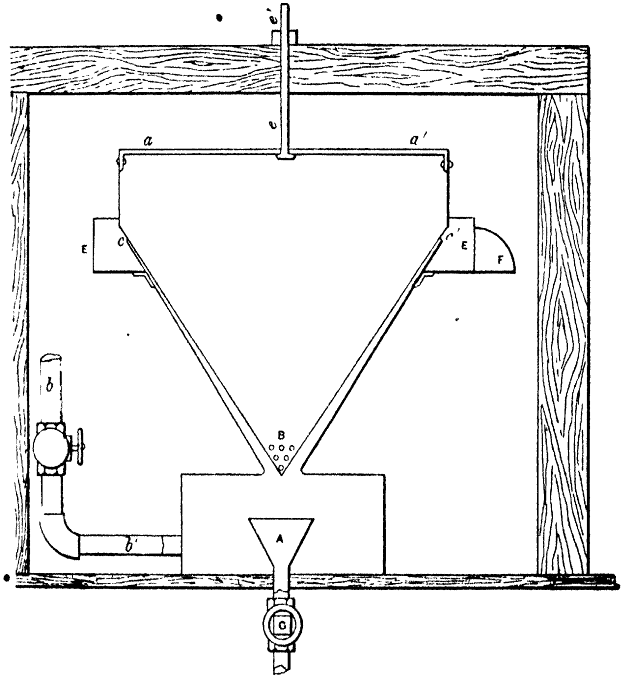

Fig. 194 Colorado classifier.

Single machines working on the principle of the spitzlutte are used for the purpose of getting a classified material for further treatment. Among these may be named the **Colorado classifier** described by H. E. Armitage[1]. It consists, as shewn in Fig. 194, of two inverted sheet iron cones suspended one inside the other. The pulp to be classified runs into the inner cone, whence it escapes through the perforations *B* near its apex and meets an upward current of clear water from the pipe *bb'*; the finer slimes are carried up between the cones and escape at *cc'* into the circular trough *EE'* and thence through the spout *F*. The coarser mineral falls into the hopper *A* and is drawn off as required through the valve *G*.

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[1] *Trans. Amer. Inst. Min. Eng.*, "Concentration of Low Grade Ore," by H. E. Armitage, Vol. XVIII. 1887, p. 257.

The **Hancock**[1] **classifier** used at the Moonta mines in Australia and others is very similar in construction and mode of action. It consists of an inverted cone 8 feet high and 8 feet in diameter at the top; there is a vertical shaft in the centre making about 20 revolutions per minute to which are keyed paddles 9 inches wide, and of such a length as to leave a 3 inch clearance between them and the sides of the cone. At the apex of the cone a small stream of clear water under pressure is introduced. There is an outlet valve at the bottom and others are fixed up the side of the cone at intervals of 6 to 8 inches apart, from which pulp of different degrees of fineness can be discharged as required. This machine will classify about 35 cubic feet of ordinary pulp per minute.

Another similar machine is the classifier used at the Frongoch mine, Cardiganshire[2].

B. APPLIANCES FOR FINAL TREATMENT.

These are generally applied to coarser material than the previous group; the products may occasionally be subjected to further treatment, but the object is not that of classifying mineral, as in the appliances just considered, into a number of classes, each of which is to be further treated, but to separate the mass of mineral into two classes, one worthless, the other valuable, one at least of which therefore requires no further treatment, the operation being in this sense a final one. Among these machines may be mentioned the Robinson coal washer, the "syphon" washer of the Mechernich mine, and the diamond washing pan.

The Robinson coal washer in general arrangement is very like the machines last described. It is shewn in Fig. 195. It consists of a sheet iron cone placed apex downwards, in the axis of which revolves

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[1] *Report on the Loss of Gold in the Reduction of Auriferous Veinstuff in Victoria*, by Hy Rosales, p. 61.

[2] *Ore and Stone Mining*, C. Le Neve Foster, 4th edn. p. 588.

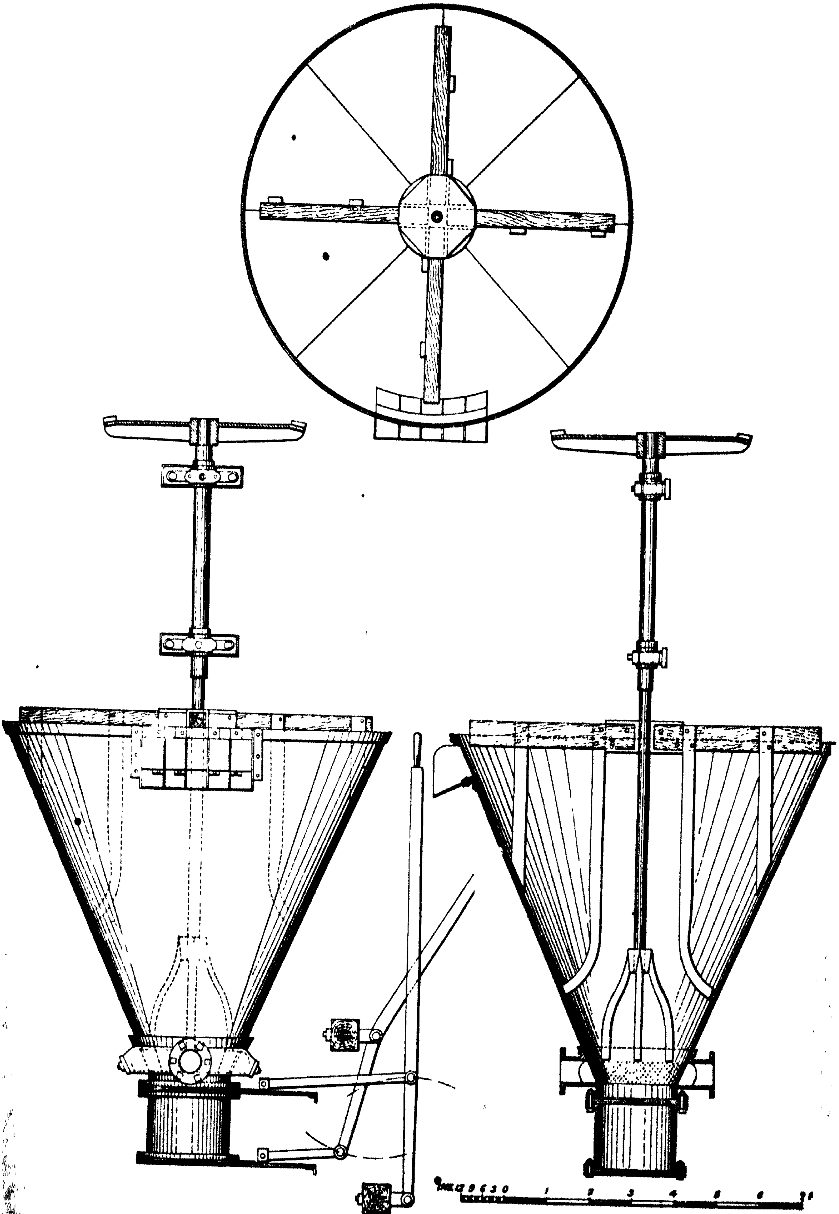

Fig. 195. Robinson coal washer. Elevation, vertical section, and plan.

a vertical shaft carrying four arms, to which stirrers are attached. The lower end of the cone is closed by double sliding doors worked by levers; the upper door is open and the lower one closed when the washer is at work. Round the lower part of the cone is an annular pipe, perforated on the inside so that jets of water rise up from it all round into the cone. This pipe is fed from a water tank situated some 30 feet above the level of the washer. The small coal (which has passed through a screen of about ¾ inch mesh) is fed into a hopper, from which it runs into the washer through a shoot with a sliding gate, by means of which the rate of supply is regulated. Water from the high-level tank is run into the cone, and the small coal is carried round in the water which is kept in swirling motion by the revolving stirrer; it tends to sink, but is carried upwards by the ascending water currents from the annular pipe, so that only particles of heavier minerals (shale and pyrites) can sink down into the space between the two sliding gates at the bottom of the cone. The coal makes about three-fourths of a complete revolution suspended in the water, and is then discharged at the overflow over a fine screen, on which it is drained from most of the accompanying water; the latter runs into a settling pit, whence it is pumped up again by a pulsometer into the high level tank. The coal drops into waggons and is sent to the coke ovens; the finer slimes are separated out in the settling pits. When the space between the two sliding gates is nearly filled with shale and pyrites a small car is wheeled into position below the cone, the upper gate is closed and the lower one opened, thus allowing the heavier minerals to drop into the car, by means of which they are wheeled away to the waste dump. The lower gate is again closed and the upper one opened, and the operation continues as before. The water circulates continuously, a sufficient amount being added to make up for loss.

The general arrangement of a single-washer plant is shewn in Fig. 196, in which *O* is the washer, the vertical shaft of which is worked by the engine *K*. The dirt falls into the chamber *L* at the bottom of the washer, and can be dropped thence into the waggon *M*. The small coal is supplied from a storage hopper, and runs through the shoot *E* into the washer. The washed coal is discharged through the spout *H*, the bottom of which is finely perforated, into the hopper *J*, and thence as required into the waggon *N*. The water that drains from this washed coal collects in the tank *I*, whence the pulsometer *F* pumps it through the pipe *D* into the cistern *A*, which is fitted with an overflow pipe *C*, whilst the pipe *B* takes the water down to the washer.

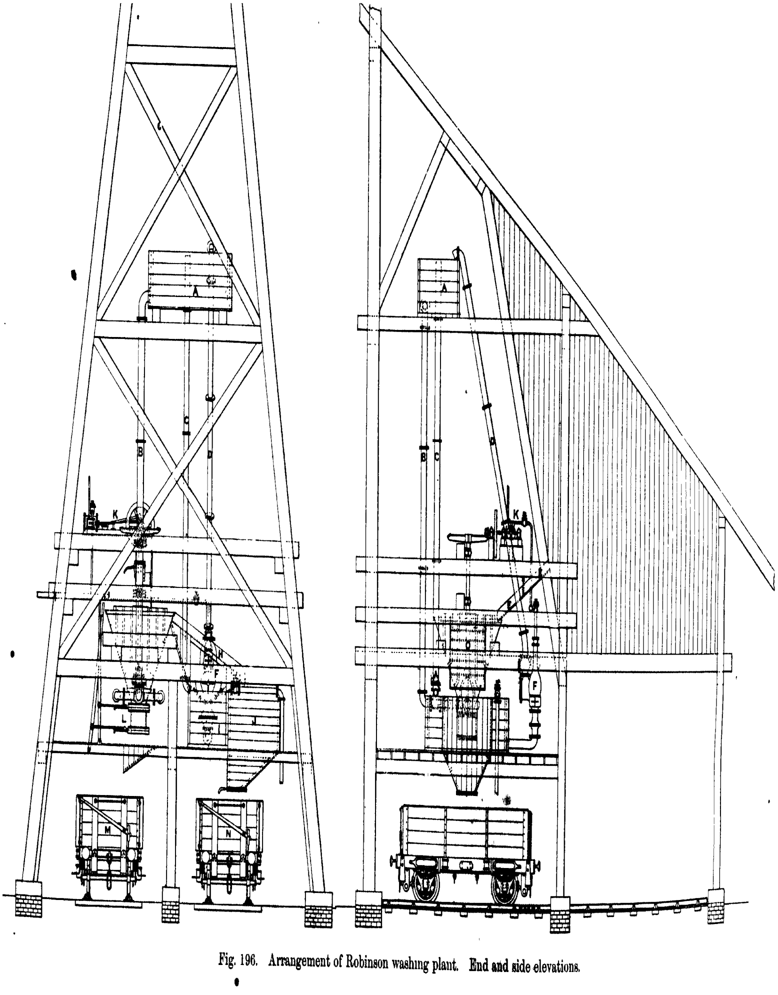

Fig. 196. Arrangement of Robinson washing plant. End and side elevations.

One man and one or two lads can attend to one of these machines, which can treat 150 to 200 tons per day.

At the South Derwent colliery one of these washers treats 180 tons of small coal per day. The coal contains 10 to 11 per cent. of ash, which is reduced by washing to 4 to 5 per cent., the sulphur present being also diminished by ½ per cent.

At the Barrow colliery, near Barnsley, a Robinson washer treats 24 to 28 tons per hour; it takes out dirt equal to 8 to 10 per cent. of the total weight of small coal treated. The cost of crushing and washing is 2½*d.* per ton, and it is said that "the coal being reduced to a uniform size, previous to washing, the Robinson machine gives fairly satisfactory results[1]."

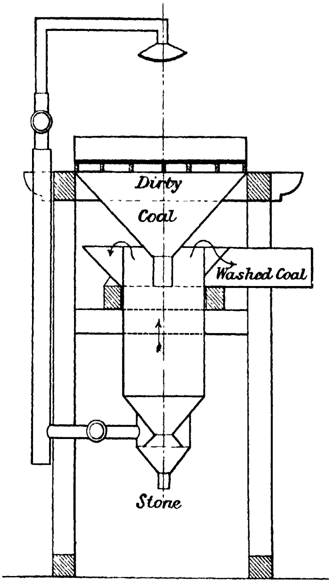

Fig. 197. Ampsin coal washer.

A still simpler form of coal washer was that designed by M. Dor at Ampsin[2], shewn in Fig. 197, which gave only moderate results, and at the same time had but a small capacity, namely 4 tons in 10 hours.

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[1] *The Min. Inst. of Scotland*, "Report of Committee on Coal Cleaning," Vol. XI. p. 162

[2] *Ann. des Mines* (Paris), "Préparation mécanique des minerais de Plomb, etc.," by A. Henry, Ser. 6, Vol. XIX. 1871, p. 352.

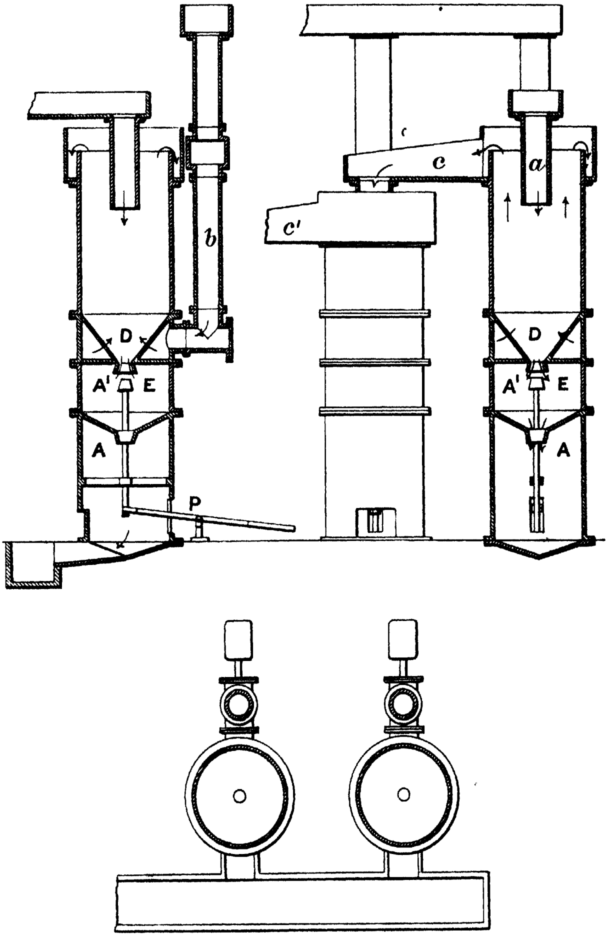

Fig. 198. Mechernich syphon washer. Plan and sectional elevations.

The so-called **Syphon-washer**[1] (*Heberwäsche*), used at the lead mines of Mechernich, is shewn in Fig. 198, its principle depending, in spite of its name, in no wise on that of the syphon. As will be seen, these machines are worked in pairs, consisting of two quite identical machines. The crushed ore (*Knotten-Sandstein*), consisting of sandstone with small nodular concretions of galena, forming about 2¼ per cent. of the whole, is introduced in a current of water through the pipe *a*; it falls in the cylindrical body and meets an ascending stream of hydraulic water, which is introduced through the pipe *b*, and ascends through the meshes of the stout conical sieve *D* that forms the false bottom of the cylindrical body. The heavy particles of impure galena fall against the ascending current and accumulate in the lower chamber *E*, whence they are emptied out from time to time by depressing the lever *P*, which opens the lower plughole *A*, the upper plughole *A′* being at the same time closed to prevent the water escaping from the main body of the apparatus, and its regular working being thus interfered with. The overflow from the first machine passes through the spout *c* into the second, the action of which is identical; the overflow from the second through the spout *c′* is barren sand and runs to waste. It is said that these machines could treat 1500 to 1600 tons of crushed ore in 24 hours, with however a very heavy water consumption amounting to about 280 cubic feet of water per minute.

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[1] *Ann. des Mines* (Paris), "Préparation mécanique des minerais de Plomb, etc.," by A. Henry, Ser. 6, Vol. xix. 1871, p. 354.

This apparatus has been improved by Mr Osterspey, who has added a very ingenious arrangement for making the intermittent discharge automatic. The self-discharging "syphon-washer" is shewn in Fig. 199[1]; it is rectangular in plan and is made of boiler plate, being carried upon a pair of steel girders; *B* is the main body of the appliance in which separation takes place, the crushed ore being brought in by a stream of water at *G*, and the overflow taking place at the opposite side. The hydraulic water enters the main body, as before, through the perforated conical bottom *b*, the large central opening in which is closed by a plug attached to a rod *e*. This rod is worked by a lever *h*, pivoted at *i*, the other end being carried by a rod *f*, attached to a float *S*; the position of *h* on the rod *f* is adjustable. The hydraulic water enters through the pipe *a* into a rear chamber *E* communicating with *B* as shewn. The upper part of this chamber *E* is divided into a front closed portion *u*, into which the hydraulic water enters, whilst the back portion *C* is only closed by the float *S*.

The mode of action is as follows: the pulp of crushed ore enters the body *B*, where it meets the rising stream of water, which carries up the lighter portions that escape with the outflowing stream of water. The

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[1] *Berg. u. Hütt. Ztg.* 1886, p. 476.

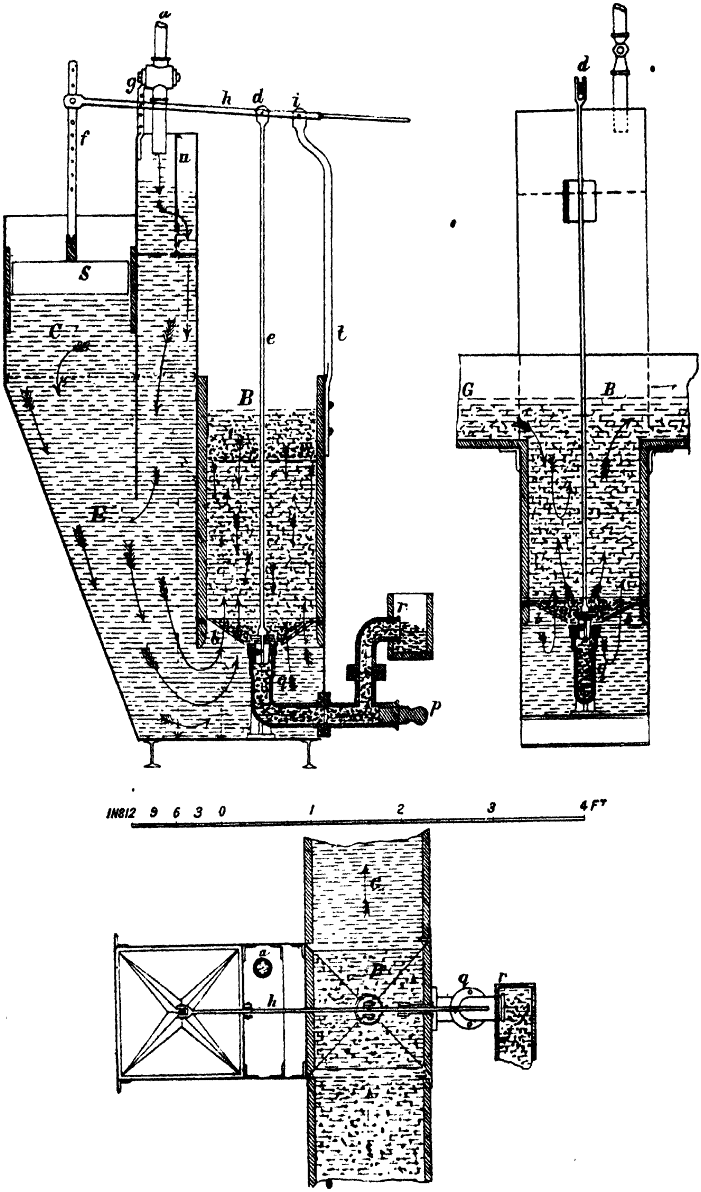
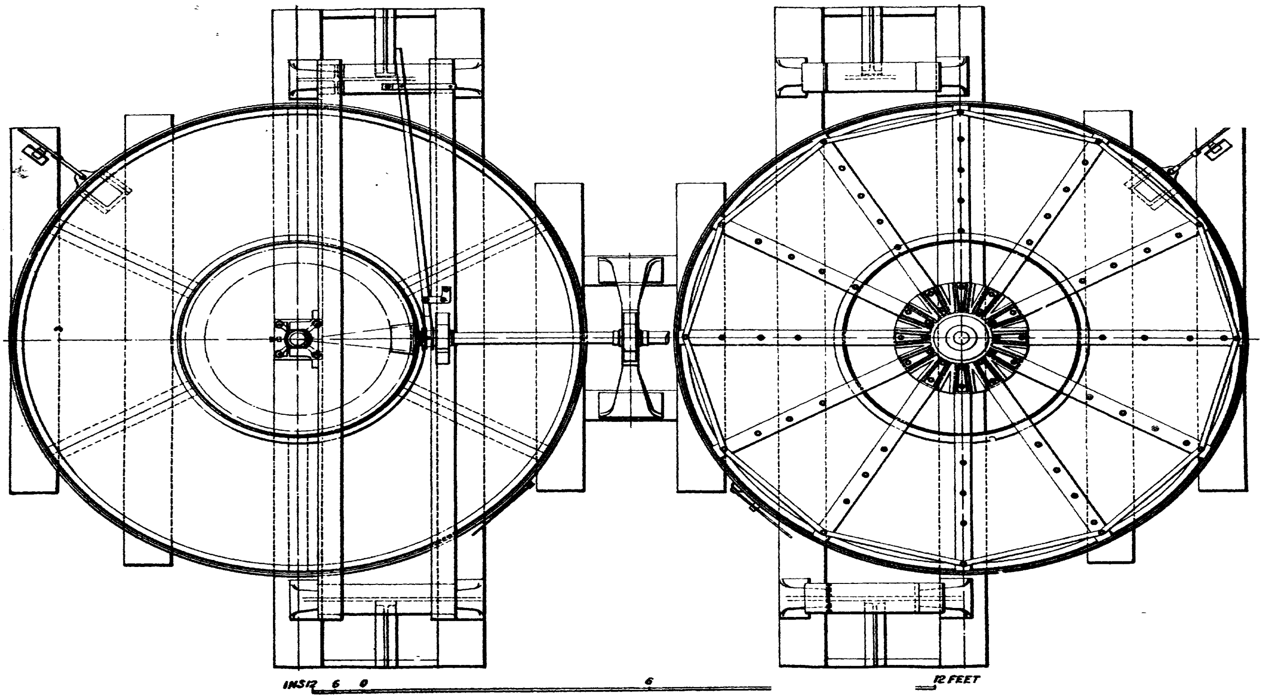


Fig. 199. Automatic syphon washer. Plan, longitudinal and transverse sections.

heavier particles of ore sink and accumulate on the perforated bottom *b*. In proportion as they accumulate they choke the perforations in *b*, and thus bank back the hydraulic water in *C*; the float *S* is thus lifted, and this in turn, by means of the combination of levers shewn in Fig. 199, lifts the plug, and allows the accumulated ore to escape through the pipe *q* into the trough *r*. The level of the water in *C* rapidly falls, the float *S* sinks with it and again presses the plug into the central aperture of *b*; the first condition of affairs is thus restored, and the operation goes on as before. These machines are arranged in series, usually of three. Each machine has an hourly capacity of 12 to 14 tons of material crushed below ¼ inch, with a water consumption equal to 26 cubic feet of water per minute.

The **Rotary Diamond Washer** is used for washing the weathered blue ground of the Kimberley mines; the blue ground is washed by a stream of water into a trommel, about 2 to 3 feet in diameter and 6 to 8 feet long, with wire gauze or perforated sheet iron mantle, the mesh being 1 to 1¼ inches. The oversize from the trommel, known as "cylinder lumps," is returned for further weathering; the pulp passes to the Rotary washing machine, shewn in Fig. 200. This consists of an annular pan, the outer diameter of which is 12 to 18 feet and the inner diameter about 6 to 10 feet, the width of the annular pan being thus 3 to 4 feet. The outer rim is 12 to 24 inches high, the inner rim 6 to 8 inches. The centre of the open inner space is occupied by a vertical shaft carrying 10 radial arms, each of which is fitted with 6 to 7 knives or teeth, set vertically downwards, their points coming within about ½ inch of the bottom of the pan; they are arranged in a spiral so that when the shaft is rotated (at about 10 revolutions per minute) the teeth tend to carry everything towards the outer circumference of the pan. The pulp enters the pan through a slot in the outer rim and escapes over a depression in the inner rim; usually the escaping pulp flows into a second similar machine, and thence into the tailings pit. The tailings are raised by elevators of various kinds to the top of heaps some 30 feet high, where the semi-solid mud is deposited, whilst the muddy water, screened off from it, runs back to the trommels. The heavier portion of the mineral contents, including the diamonds, remains on the bottom of the pan, whence it is removed every day for further treatment. Each such machine treats about 300 tons of blue ground in the 10 hour shift, the concentrated mineral amounting to about 3 tons. A plant,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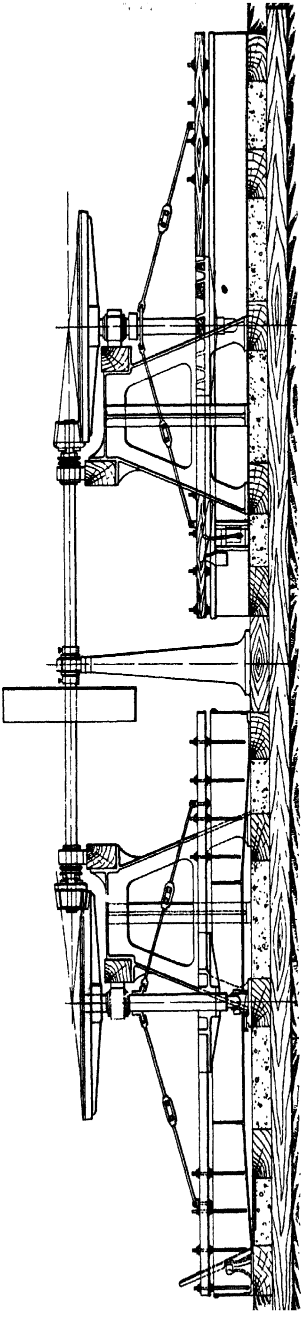

Fig. 200. Rotary diamond washer. Plan, elevation and section.

consisting of trommel, two pans and an elevator, takes about an 8 Nom. H.P. engine (say 24 I.H.P.) to work it.

This washing machine is thus a separator of the ordinary type, the most noteworthy point about it being the use of thick muddy water instead of clear. It has to effect the separation of minerals not differing greatly from each other in specific gravity, and it has already been shewn how small differences of density can be accentuated by the use of fluids of higher specific gravity than water; the finely divided mud suspended in water practically acts as a fluid of increased specific gravity, and thus promotes the separation of minerals which could only be effected with difficulty in clear water.

The tossing kieve or dolly-tub is an appliance that should be classed here; the former is the name given to it in Cornwall, the latter in the north of England and Wales. It is used for cleaning the concentrates produced by the buddles (see pp. 293 to 299) in the treatment of both tin and lead ores. In its simplest form it consists of a stout tub, usually about 3 feet high and 3 feet in diameter, strongly hooped with iron. The tub being about three-quarters full of water, one man shovels in gradually the "heads" to be cleaned, whilst another keeps the contents in circular motion by stirring with a long-handled shovel. When the tub is about half full of mineral, the first man strikes a succession of blows upon the side of the tub, usually by means of a heavy bar of iron, the lower end of which rests on the ground. As soon as this tapping commences the other workman leaves off stirring and withdraws his shovel, the tapping being steadily continued. Under this jarring action the material settles rapidly, the clean heavy mineral at the bottom, with a layer of the lighter sands on top. The upper layers are scraped off with a suitable scraper, a trowel being often used, and the concentrated mineral in the bottom of the tub can then be shovelled out.

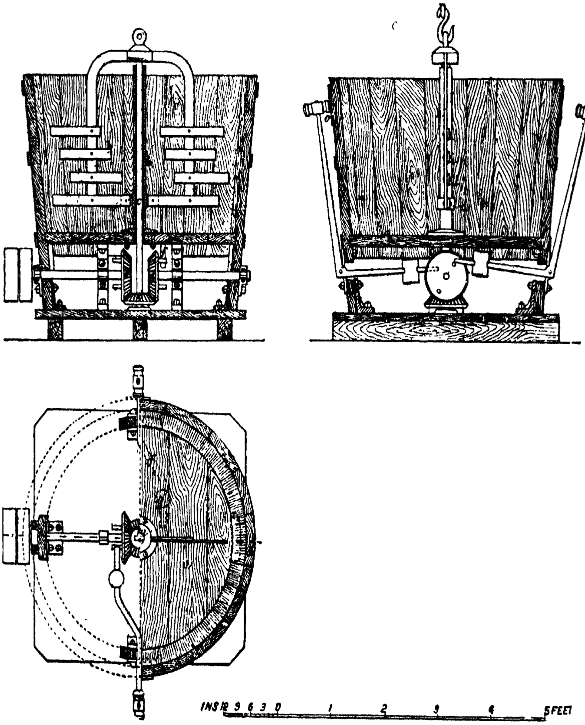

Fig. 201. Mechanical dolly-tub. Plan, vertical section and elevation.

In Cornwall the first portion of the above operation is known as "tossing," and the second as "packing"; when performed, as it sometimes is, with a tub lying at an angle and not standing upright, the operation is called "chimming."

Mechanical dolly-tubs are at times used in which the stirring or the packing or both are performed by machinery. A good form is shewn in Fig. 201[1], as used in California. It consists of a tub, 2 feet 6 inches deep, 3 feet in diameter at the bottom and 4 feet at the top. Up the centre runs a 3 inch cast iron pipe, within which revolves a vertical spindle driven by bevil spur gearing as shewn; to the top a yoke can be keyed carrying stirrers; this can easily be disconnected and hoisted out of the tub when the period of packing commences. Two trip hammers are made to act upon opposite sides of the tub, being caused to strike a sharp blow by means of the weights attached to the horizontal arms of the bell-crank levers, the vertical arms of which carry the hammer heads. The hammers are tripped by means of two pins set in the face of one of the spur wheels. During the period of stirring, wedges are pushed in between the shafts of the hammers and the sides of the tub, which lift the ends of the levers clear of the pins. The shaft is made to revolve 48 times per minute. There are several modifications of this arrangement in use, but dolly-tubs are now but little used.

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[1] *Engineering*, April 22nd, 1881, p. 404.

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CHAPTER VII.

JIGS.

In the appliances classed as Jigs[1], particles of mineral are subjected to alternately ascending and descending vertical currents of water, each acting for a very brief period; in practice intervals of rest generally occur between successive impulses, but this fact in no wise affects the mode of action of the apparatus. Assuming two equal-falling particles of different specific gravities to be placed upon a screen, the mesh of the latter being smaller than the diameter of the particles, and, further, allowing these particles to be subjected to an upward moving current of water, the velocity of which is superior to the ultimate falling velocity of the particles, it is evident that both particles will be lifted. It has already been shewn (p. 223) that under these conditions both particles will ultimately attain the same ascending velocity, but that at the outset the specifically lighter of the two particles will be lifted the faster. If the current is interrupted and reversed before the régime of equal velocity is established, the former particle will be above the latter. If these same two particles are submitted to a descending current, the specifically heavier particle will commence to fall the more rapidly, hence the action of the descending water current will reinforce that of the ascending current, provided that the action of each is continued for only a brief period of time. If instead of only two particles a large number of particles of both kinds were placed upon the screen and submitted to similar water currents, the final result would obviously be to produce two layers, a lower layer of the specifically heavier and an upper layer of the specifically lighter particles. This action would be further intensified by the fact that the heavier particles, being the smaller of the two, can slip through interstices between the particles through which the specifically lighter could not pass, and are thus able to descend with greater ease and so to reach the bottom, even more rapidly than is due to their higher falling velocities. These considerations explain the well-known fact that in practice it is easy to separate equal-falling particles by jigging, and that successful jigging can be done on particles of different sizes when the proportion of the largest to the smallest exceeds greatly the sieve ratio that would be deduced from the laws of equal-falling (see p. 220), and affords a typical example of the effects due to hindered falling. If the diameter of the particles be less than the mesh of the sieve, the particles will be able to pass through it, and if the action were continued long enough, all the particles might pass through it, unless the alternations of the currents were too rapid to admit of the lighter particles ever reaching the surface of the sieve. In any case it would be possible to stop the operation after the heavier particles had passed through, and if the surface of the sieve were covered by a layer of particles too large to pass through the meshes of the sieve, this effect would be accentuated, and it would obviously be easy so to arrange matters that all the heavier particles could pass through this layer of larger particles (known in practice as the "bed") and all the lighter particles remain above it. This method of jigging with a sieve of larger mesh than the diameter of the particles has sometimes been called the English system of jigging, and sometimes the Harz system. The two methods may be distinguished by the respective phrases "jigging through the sieve" and "jigging over the sieve." The former method is usually applied to finer material, the latter to coarser, so much so that the German terms for jigs working on the respective principles correspond to "sand jigs" and "coarse grain jigs." The limit for jigging through the sieve is in some cases, however, as high as ⅜ inch. If the appliance be so arranged that a series of particles stream across such a screening surface, and are during their passage submitted to a jigging action, it is possible to so proportion the conditions that the period of flow across is just sufficient to allow the heavy material to pass through the sieve, whilst the lighter ones would simply flow over it.

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[1] In the North of England Jigs are sometimes called hotching machines; Jigs used for washing coal are often called bashes.

The alternating water currents may be produced in two ways, either by moving a sieve up and down in a tank of water, or by having a sieve fixed in a tank, the water being caused to rise and fall by means of the reciprocating action of a piston or by some similar device. The former was the older method and continued in use for centuries, until the fixed sieve jig was introduced in Cornwall by Captain Petherick about the year 1830[1]. The tank in which the sieve is placed is known as the "hutch," and any mineral that accumulates in it is called "hutchwork." Either the fixed or the movable sieve may be employed by jigging either through or over the sieve. The mineral to be jigged is usually brought on to the screen suspended in water, so as to form a pulp; in addition hydraulic water is usually supplied either over the sieve or into the hutch; in the former case the downward action of the water current, or the action of suction, as it is termed by Prof. Richards, in the latter case the upward action (Prof. Richards' pulsion) is usually intensified. In the latter case the particles of mineral on the sieve are kept in a looser condition, or are livelier, and more readily acted upon by the water currents. These water currents also play in many cases an important part in transporting the particles of the lighter mineral (more rarely the heavier) horizontally over the sieve and thus carrying them out of the jig, and, as already shewn, the rate of flow of such a current determines the time during which the mineral is exposed to the jigging action, and thus can be made to modify the results produced by any given jig.

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[1] *Trans. Inst. C. E.* Vol. xxx. 1869-70, p. 125.

A notable improvement was made by M. Berard about 1850, in jigs used for washing coal, by fitting the sieves with movable gates, so that waste and concentrates can be continuously discharged in any desired proportions. In coal washing it will be noted that the lighter portion, usually spoken of as the waste when ores are jigged, is really the valuable portion, namely, the clean coal, whilst the heavier concentrates constitute the useless dirt, shale and pyrites, which it is desired to get rid of. A still greater improvement consisted of combining several individual jigs into one compound jig, two-, three-, four-, and five-compartment jigs being used, though the first and last are rare. These jigs are so arranged that the whole of the waste from the first jig passes on to the second, the waste from the second on to the third, and so on until the clean waste is discharged from the last jig. The first jig takes out the heaviest concentrates, either through the sieve or over it from a suitably arranged discharge. The concentrate made by each individual jig is always the heaviest portion of the pulp supplied to it, so that a series of concentrates of gradually decreasing specific gravity can be produced. A compound jig (with fixed sieves) thus arranged is often spoken of as a Harz[1] jig, owing to its having been first introduced in Clausthal.

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[1] Prof. Richards in his *Ore Dressing* (Vol. I. p. 507) restricts the term Harz jig to piston jigs as here described, in which "a plunger receives its up and down motion from an eccentric revolving at a uniform rate," and states that this is the definition commonly accepted in the United States of America.

It will be noticed that the conditions of jigging can be varied through a very wide range, and that these appliances can accordingly be adjusted with very great delicacy.

The following conditions all influence the mode of action of a jig:

1. The quantity of material treated, its density, its size, and the ratio between the dimensions of the largest and smallest particles (i.e. the sieve-scale according to which it has been sized).
2. The water supply, both as regards the proportion of water in the pulp, and the amount of hydraulic water, and the point of its introduction.
3. The rate of the oscillations of the water current and the amplitude of such oscillations; also the mode in which these oscillations are produced, whether with a fixed or a movable sieve.
4. The relative speeds of the upwards and downwards currents in each oscillation.
5. Whether jigging is performed over the sieve or through the sieve; in the latter case, the depth of the bed, and the nature and size of the particles composing it.

Both the speed of the oscillations and their amplitude may be varied within wide limits in the first place, and in the second place their character may be altered, inasmuch as the upward and downward movement may both be performed at the same rate of speed, or one—usually the upward movement—may be made the more rapid of the two, thus making the period of suction longer than that of pulsion. According to Prof. Richards[1] suction is more efficient than pulsion in jigging unsized material, whilst pulsion is more efficient than suction in treating closely sized products. According to G. G. Bring[2] the ascending current is the more effective in coarse-grain jigs, and the descending in fine-sand jigs; he holds that the bed in the latter class acts as a series of narrow channels in which the ascending and descending currents move. The vast majority of jigs in ordinary use employ a uniform velocity for both upward and downward impulses. The finer the material that is being jigged, the shorter and the more rapid must the alternations of current be, for the obvious reason that the finer the particles the sooner is the equal-falling régime attained. The following table of these data is compiled from Linkenbach's well-known work; it applies especially to the treatment of lead and zinc ores in a gangue of quartz and schist, but may fairly be taken as generally applicable; it refers, as will be seen, to jigging over the sieve.

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[1] *Ore Dressing*, Vol. I. p. 631.

[2] *Jern-Kontorets Annaler*, "Experimentella studier öfver sättmaskinens verkningsätt," by Gust. G. Bring, 1906, p. 321.

Diameter of particles	Number of oscillations per minute	Depth of oscillations	Horse-power* required	Quantity treated* per 10 hour shift	Nature of sieves	Mesh of sieves	Life of sieves in 10 hour shifts	Water* consumption per minute
inches		inches		tons		inches		gallons
1·2—0·8	110—120	3	1·5	15	Punched sheet steel 0·2 in. thick	0·4	450	33—40
0·8—0·5	,,	2·4	,,	12·5	,, ,, ,, 0·12 ,,	0·24	300	,,
0·5—0·3	,,	2	1·25	10·5	,, ,, ,, 0·08 ,,	0·16	250	,,
0·3—0·2	130	1·7	,,	9	Brass wire	0·12	100	26—33
0·2—0·12	,,	1·4	,,	8	,,	0·08	90	,,
0·12—0·08	140	1	1·0	7·5	,,	0·06	75	,,
0·08—0·06	,,	0·6	1·0	7·2	,,	0·04	50	,,
Below 0·06	150—180	0·45—0·6	0·6	4·5—6·5	,,			

* **These data apply to a compound jig comprising three sieves; the capacity and water consumption of a single jig will be about the same as those given above, because the same pulp flows successively over each of the three sieves (see p. 258), but the horse-power required for a single jig will only be one-third to one-half of the figure given in the table.**

The following table shews the proportions recommended by Mr Thomas Sopwith[1] on very similar ores, but representing considerably older practice:

Diameter of particles	Number of oscillations per minute	Depth of oscillations	Quantity treated per 10 hour shift
inches		inches	cwts.
0·4 —0·28	96	2¼	244
0·28—0·2	86	2	220
0·2 —0·1	84	1¾	112
0·1 —0·06	82	1	71

Commans[2] has given a table, based apparently on that of Linkenbach, as follows:

Size of particles	Length of stroke	Number of strokes per minute
inches	inches	
1·18—1·77	3·15—3·94	100—110
0·79—1·18	2·36—3·15	110—120
0·51—0·79	1·97—2·36	110—120
0·31—0·51	1·57—1·97	110—120
0·20—0·31	1·18—1·57	120—140
0·12—0·20	0·79—1·18	120—140
0·08—0·12	0·59—0·79	140—180
0·06—0·08	0·39—0·59	140—180

He states that in jigging over the sieve, the mesh for the finer sizes should be a size or two smaller than that of the particles, and for sizes above 0·3 inch, the sieve holes may be half the diameter of the particles. He gives the power required for a three-compartment jig treating the finer sizes as 1 H.P. and coarser sizes 1½ H.P., and the water consumption for such a jig as about 30 to 40 gallons per minute; he states that the capacity of an ordinary jig with sieve 2 feet 6 inches to 3 feet long and 18 to 20 inches wide is:

20—80 cwt. for sizes between 0·51 and 0·79 inch diameter
15—20 „ „ 0·12 „ 0·51 „
10—15 „ „ 0·06 „ 0·12 „

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[1] *Proc. Inst. C. E.*, "The Dressing of Lead Ores," by Thomas Sopwith, junior, Vol. xxx. p. 106.

[2] *Proc. Inst. C. E.* 1893-4, Vol. cxvi.

Prof. Richards[1] gives elaborate tables which shew that the speed of jigs in America considerably exceeds the figures given above. The following table is compiled from his data:

Diameter of particles	Average number of oscillations per minute	Average depth of oscillation
inches		inches
2·5 --1·25	129	2·6
1·25-- 0·63	131	2·0
0·63--0·31	144	1·4
0·31- 0·16	176	0·9
0·16-- 0·08	235	0·55
0·08 - 0·04	250	0·48
Below 0·04	281	0·19

The following table shews the practice of E. Ferraris[2] at Monteponi, Sardinia:—

Diameter of particles	Number of oscillations per minute	Length of piston stroke	Horse-power required	Quantity treated per 10 hour shift	Mesh of sieve	Clear water consumption per minute
inches		inches		tons	inches	gallons
0·8 --1·2	100	1·6—2·0	1·25	5	0·4	3·1
0·55—0·8	110	1·4—1·8	1·1	4·5	0·32	22
0·4 —0·55	120	1·2- 1·6	1·0	4	0·24	16·5
0·28—0·4	125	0·8—1·4	1·5	3	0·4	11
0·2 —0·28	180	0·8 - 1·2	1·5	3	0·32	10
0·14--0·2	150	0·6- 0·9	1·5	3	0·24	8·75
0·08—0·14	180	0·6—0·8	1·5	3	0·16	8·75

The first three jigs in this table are two-compartment jigs, each sieve being 18 by 30 inches; the other four are five-compartment jigs, each sieve being 18 by 20 inches; the former work over the sieve, the latter through the sieve, the bed consisting of iron discs (the waste from punching machines), the diameter of which is about 50 per cent. greater than that of the holes in the sieves. The ores treated are lead and zinc ores.

Modern jigging practice is fairly represented by the diagram, Fig. 202, which represents the author's average practice, and may be taken as generally applicable, and from which the data required may be readily scaled off.

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[1] *Ore Dressing*, Vol. I. pp. 593–595.

[2] *Trans. Amer. Inst. Min. Eng.* 1908, p. 363.

Jigs may be used on material ranging in size from 3 inches down to 0·01 inch.

When jigging through the sieve, the mesh of the sieve must, of course, be considerably greater than the diameter of the particles that have to pass through it; the material of which the bed is composed, its specific gravity, and the size of the particles, as also the depth of bed, are very important. The material must evidently not be too soft or it will suffer from abrasion; it must be considerably heavier than the waste to effect a good separation, and is often of the same material as the concentrates, as then anything abraded off it goes into the hutchwork and does no harm. Thus in dressing galena the same mineral is often used for the bed; iron pyrites is often used for a bed; lead shot, iron punchings, and cast iron shot have also been used. All these answer well, but metallic iron is apt to rust when the jigs are idle, especially

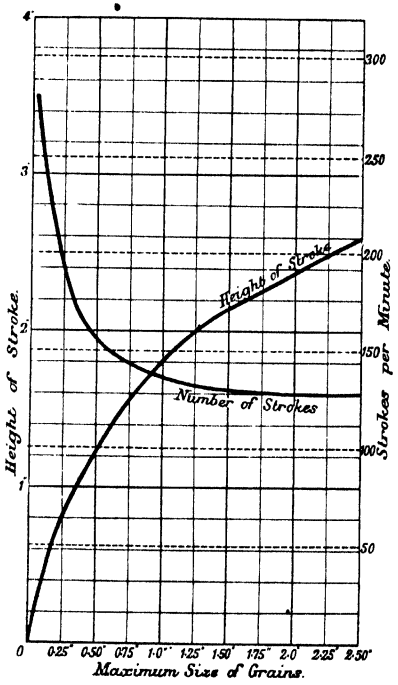

Fig. 202. Diagram of jigging practice.

when sulphuretted ores are being dressed. In jigging fine coal through the sieve a bed of cleaved pieces of felspar is almost always used; this is fairly hard and of a suitable specific gravity, as being considerably heavier than clean coal, but its special advantage lies in the shape of the fragments. Felspar cleaves into pieces of the cross-section shewn at *abcd*, Fig. 203, in which *XY* represents the sieve surface upon which the piece of felspar is lying. Under the influence of ascending water currents, indicated by the arrows, the piece of felspar will tend to turn about the edge *a* into the position indicated by the dotted lines *ab′c′d′*, and when the current is reversed, it will tend to drop back into its original position. These pieces of felspar thus act like valves, alternately opening to the ascending water current and closing when the current descends; they thus allow pieces of shale, pyrites, etc. to drop through, but prevent any coal finding its way through the sieve. The pieces of felspar range from ½ inch to 1½ inches in length, and must have an area about 50 per cent. greater than the mesh of the screen upon which they rest. The pieces of felspar can be kept in use until the edges get rounded off by abrasion, when they must be replaced. The larger pieces can be broken down to a smaller size, but when the smallest size is worn round it must be thrown away. The life of a felspar bed under ordinary circumstances may be averaged at about six weeks; the best felspar for this purpose is obtained from Norway.

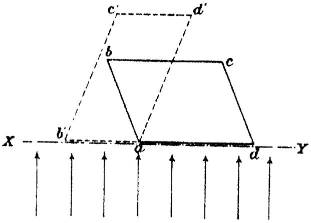

Fig. 203. Diagram of action of Felspar bed.

The hand-jig is the original type of jigging machine; in its simplest form, as a sieve jerked up and down in a tub of water, it was in use as far back as the 16th century as shewn by the quaint wood-cut, reproduced from the well-known work of Agricola, which forms the frontispiece to this book. The lever-worked hand-jig is still in use to some extent, a modern form being shewn in Fig. 204[1], whilst the mode of using the appliance is clearly shewn in Fig. 205[1]. The sieve, which is rectangular and usually about 3 feet 6 inches to 4 feet long by 1 foot 6 inches to 2 feet wide, and 9 inches to 12 inches deep, is suspended in a wooden hutch from a lever or "break-staff"; this is put in motion by means of a longer lever, the connection between the two being made by means of a stout iron pin which passes through a slot in the break-staff, and has a collar attached above and below the latter, the space between the two collars being several inches greater than the depth of the break-staff. This arrangement causes the sieve to be lowered into the hutch quite smoothly, but lifted with a sharp jerk when it commences to rise, and gives an interval of rest between successive strokes. The hutch is kept filled with water; it usually receives a small stream of water from a tap, and an overflow is arranged, so that the level of water remains constant. Care should be taken that the overflow is led into a settling tank or slime pit, as it usually carries fine mineral matter in suspension. The sieve is usually made of stout wire gauze supported upon a grid of iron bars running across the box, parallel with its shorter side, to stiffen and support the wire gauze; the mesh is usually ¼ to ½ inch, but rarely exceeds

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[1] *Proc. Inst. Mech. Eng.* 1873, Pl. 57.

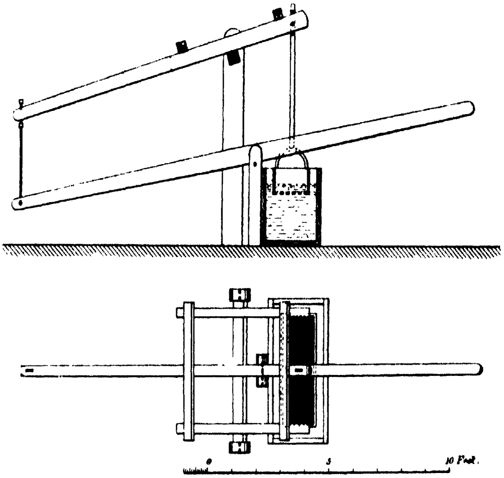

Fig. 204. Hand jig. Plan and vertical section.

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[1] *Proc. Inst. C. E.* Vol. xvii. Pl. 7.

$\frac{3}{8}$ inch. The labourer (now usually a lad) shovels the crushed ore to be treated into the sieve to a depth of about 6 or 8 inches, and then works the sieve up and down in the water by means of the long lever. For coarser material the operation of jigging is complete in $\frac{1}{2}$ to 2 minutes, for finer in 2 to 4 minutes, the speed in the former case being about 60 strokes and in the latter about 120 strokes per minute. The end of the lever is then kept up, usually by setting a prop under the end, at such height as to keep the sieve out of the water. The upper layer is then scraped off by means of a triangular piece of sheet iron or a short-handled hoe and thrown aside as waste. More ore is then thrown in and the process repeated until the worker judges that a sufficient quantity of concentrate has accumulated upon the sieve. After throwing out the top layer of waste, he then removes the next layer of middlings and puts it aside for re-treatment, and finally scrapes out the lower layer of concentrates. The hutchwork that accumulates, due either to imperfect sizing or to abrasion on the jig, is emptied out from time to time, usually once a week. It is generally clean enough to be classed as concentrates. The average capacity of such a jig may be taken at about 5 cwt. per hour per square foot of screen surface. One man or lad is required to each machine, so that the working cost is high, although the first cost is low. Such hand-jigs are now only used in dressing the ore from very small mines, in prospecting operations, or in cleaning up odd parcels at larger works; they have almost gone out of use for other purposes.

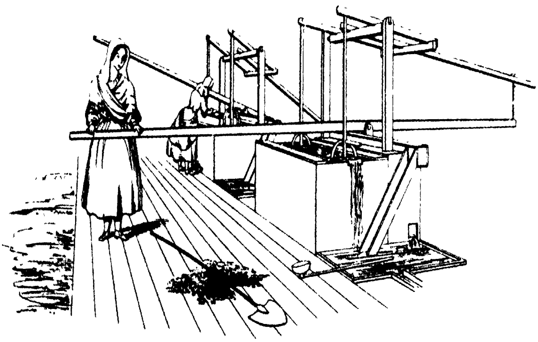

Fig. 205. Hand jig. Perspective.

In Germany, hand-jigs are also made by suspending a sieve in a hutch from an elastic spring-pole, the worker taking hold of the bar or rope by which the sieve is hung to work it up and down; the sieve and hutch are then often circular in plan. The jigging action thus produced is probably not so well adapted to its purpose as that obtained by means of the break-staff, but it is not such hard work for the labourer. In Germany, the method of jigging through the sieve is employed at times with hand sieves.

Movable sieves are at times worked by power instead of by hand; in most of these the concentrate is jigged through the sieve, whilst the tailings are discharged over the sieve, i.e. over a gate suitably placed, a stream of water being employed to keep the latter in motion.

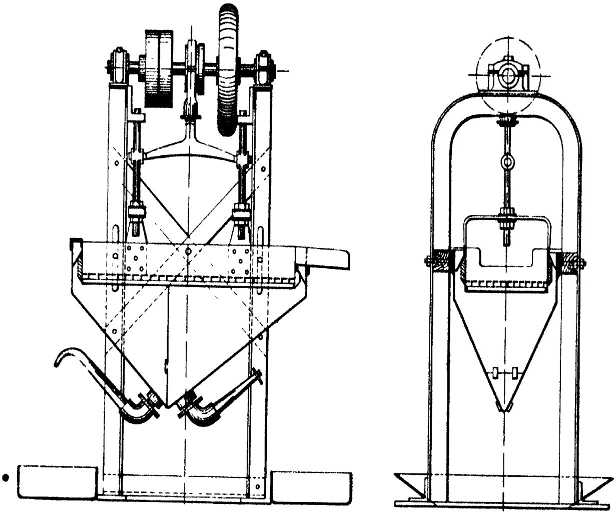

Fig. 206. Bilharz jig; single unit. Elevation and section.

A very efficient machine of this type is the machine shewn in Figs. 206 to 208, as designed by O. Bilharz[1]; a single jig is shewn in Fig. 206, and a group of four units in Fig. 207. Each unit consists of a pyramidal iron hutch in the prismatic upper part of which works a sieve, which, as shewn in the plan, Fig. 207, tapers gradually from the feed to the discharge end. The sieve is suspended by two bows from a yoke driven by an eccentric on a short overhead shaft, the upper ends of the bows being prolonged in the form of inverted Y's, the tail pieces of which form guides which constrain the sieve to move vertically up and down. The sieve is packed externally with leather like a piston, so that no water can escape between it and the sides of the hutch. Each sieve is furnished with a gate and discharge shoot, by means of which the overflow from each sieve is conveyed to the next (or in the case of the last sieve of the series is allowed to run to waste). The sieves carry a suitable bed, the machine being intended for jigging through the sieve. The concentrates accumulate in the pyramidal iron hutches; each hutch is divided into two by a transverse partition, so that if desired two qualities of hutchwork may be drawn off. The discharge for the hutch work is continuous and consists of a bent up iron pipe, so that but little water is delivered with the concentrates. The machine makes 220 strokes per minute, 0·2 inch in height. This arrangement presents the advantage of ready portability, each unit being complete in itself, and it is easy to add or take away units as required. This machine is made by the Grusonwerk Company of Magdeburg, a perspective view of whose three-compartment jig is shewn in Fig. 208. Each section of this jig weighs 815 lbs. and costs about £30.

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[1] *Oest. Zeitsch. f. Berg. u. Hütten-Wesen*, "Ueber Feinkorn- und Schlamm- Aufbereitung, etc.," by O. Bilharz, 1890, p. 213.

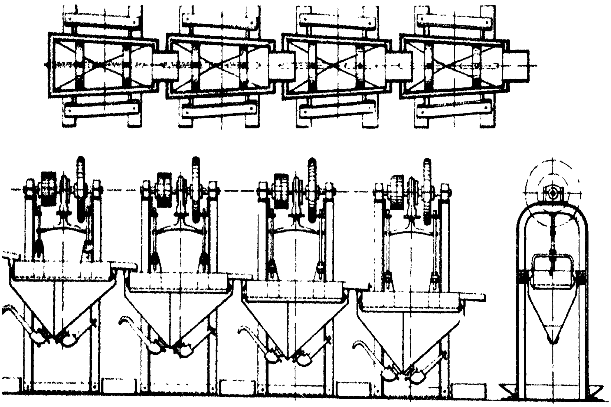
Fig. 207. Group of Bilharz jigs. Plan and elevation.

A two-compartment Felspar Washer for small coal with movable sieve is shewn in Fig. 209, as built by Messrs Sheppard & Sons, Ltd. As will be seen, its principle is practically identical with the machines last described, but it differs in matters of detail. The sieves are suspended from either end of a rocking lever and thus balance each other. The hutches are of cast iron and are furnished with screw conveyors lying in the semicircular bottom, by which the refuse is continually removed; the washed coal escapes at the outlet. This machine working at 180 to 190 one-quarter-inch strokes per minute, and using a felspar bed 4 to 6 inches deep, washes about 2½ to 3 tons per hour of fine coal, which has previously passed through a trommel with ⅜ inch mesh.

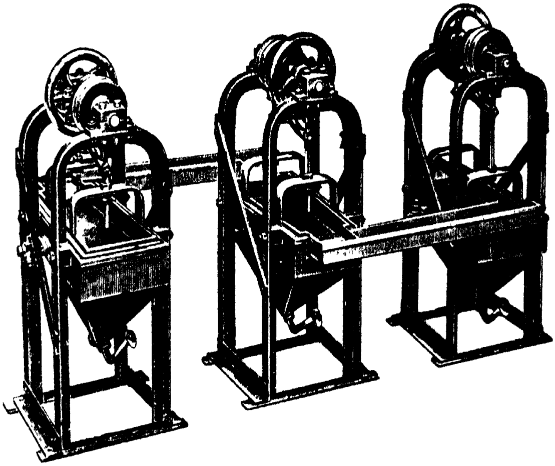

Fig. 208. Group of Bilharz jigs. Perspective.

The **Schranz jig**, specially intended for treating very fine ore or slimes, has an exceptionally long sieve; it is said to give satisfactory results.

A still larger machine is **Hancock's patent jigger**[1], used at the Moonta Copper Mines, South Australia, and some few other Australian Mines. A view of this machine is shewn in Fig. 210.

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[1] *Report on the Loss of Gold in the Reduction of Auriferous Veinstuff in Victoria*, by H. Rosales, 1895, p. 60.

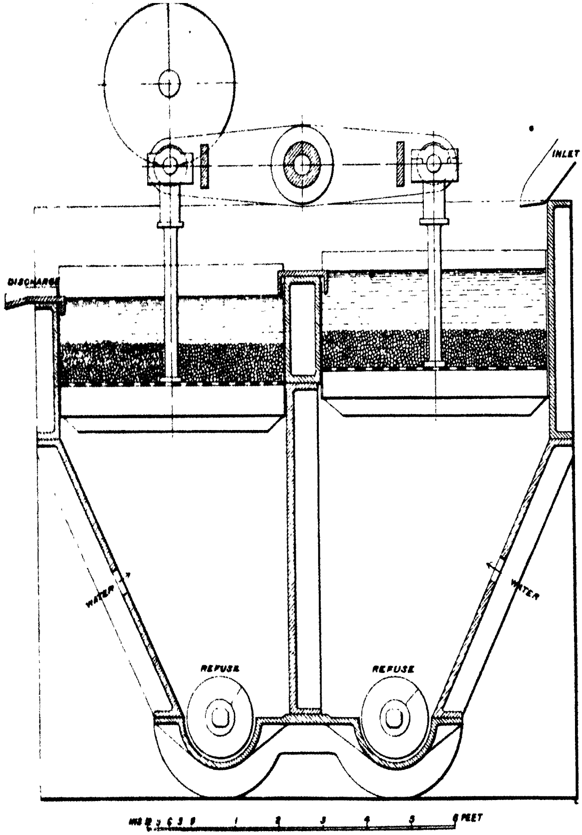

Fig. 209. Sheppard Felspar washer. Vertical section.

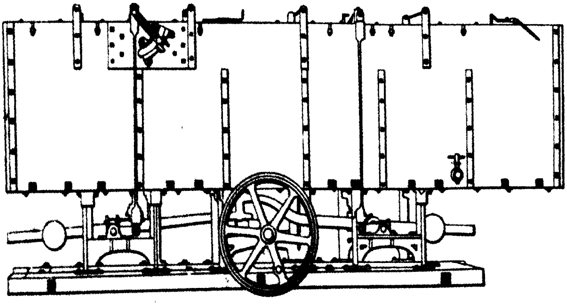

Fig. 210. Hancock jig. Side elevation.

The sieve is 20 feet in length by 3 feet 2 inches wide, and it is suspended in a hutch about 6 feet deep which is divided by transverse partitions into 6 or 7 compartments; the crushed copper ore is delivered on to the head of the sieve and is jigged through the sieve, the bed consisting of hard haematite. The cleanest ore collects in the first compartment, and progressively poorer material in the following ones. Waste is discharged at the end, where coarse middlings are also discharged over the sieve, and go into a separate receiver. The sieve has a combined up and down and forward motion, so that the waste can be moved along it with the use of comparatively little water; the throw is about ⅜ inch. The capacity of such a jig is about 7½ tons of crushed ore per hour. It has been tried also in America, but does not seem to have met with much favour there.

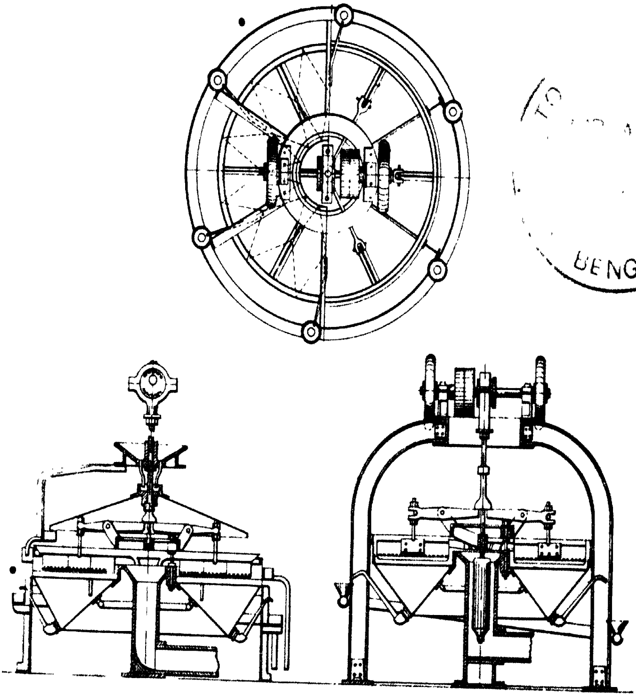

Fig. 211. Bilharz circular jig. Plan, elevation and vertical section.

DUSTRIES. BENGAL.

The **Bilharz**[1] **circular jig** is shewn in Fig. 211, in plan, elevation and vertical section; it consists of an annular sieve working in a conical hutch, the centre of which is occupied by a tube through which the waste flows off. The weight of the sieve is balanced by a weight suspended in the central tube. The hutch which is thus also annular in plan is divided radially into six compartments, the sieve being divided into a corresponding number of sections. The feed is round the outer circumference, jigging takes place through the bed, and the tailings flow out through the central tube; the machine does not make middlings. The sieve makes 200 to 220 strokes 0·2 inch high per

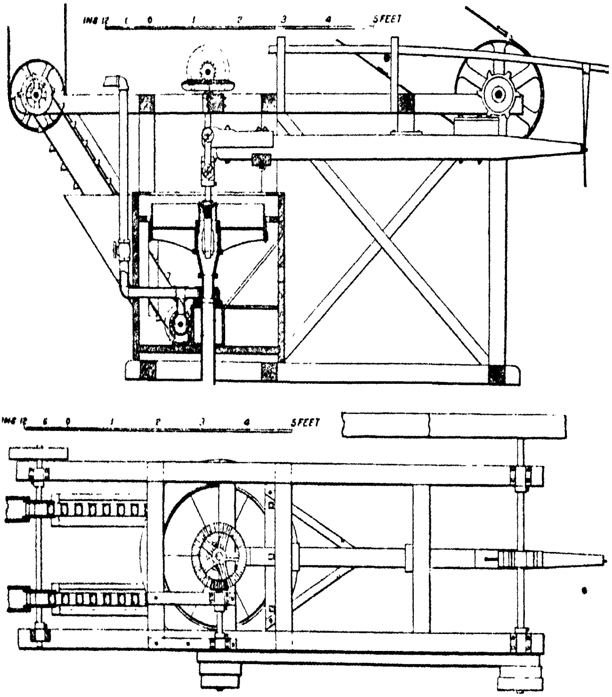

Fig. 212. Conkling jig. Plan and vertical section.

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[1] *Oesterr. Zeitsch. f. B. u. H.-W.*, "Ueber Feinkorn- u. Schlamm- Aufbereitung, etc.," by O. Bilharz, 1890, p. 216.

minute. Such a machine can treat about 12 tons of finely crushed ore, the corresponding volume of pulp amounting to 1060 cubic feet per hour, producing about 12 per cent. of concentrates.

A very similar machine is the **Conkling jig**[1] used for concentrating magnetite at Chateaugay, N. Y., shewn in plan and vertical section in Fig. 212. It consists of an annular sieve about 4 feet in diameter, working in a circular tank; the sieve is given an up and down motion by means of the long wooden lever actuated by a cam wheel, whilst the sieve is at the same time caused to revolve slowly around its axis by the bevel gearing shewn. The sieve is made of cast iron plates ½ inch thick, the holes being conical, 5/16 inch in diameter above, and 7/16 inch at the lower face; jigging takes place through the screen, a bed of ore of nut size being used. Hydraulic water enters the hutch below the sieve. The ore crushed to ¼ inch mesh is brought on to the sieve; the tailings escape through a central pipe, and the concentrates accumulate in the hutch whence they are removed by bucket elevators. The sieve makes 260 ⅜-inch strokes, having a slow up and a quick down motion; it makes also 7 revolutions per minute. The capacity of the machine is 5 tons per hour, and the water consumption 135 gallons per minute.

Neither of these circular jigs has come into extensive use.

Jigs with fixed sieves constitute by far the most generally adopted form, so much so that when the term jig is used without special qualification, this type is always meant. The motion of the water is produced in various ways, but in the great majority of instances by the reciprocation of a piston in a compartment placed by the side of that carrying the sieve, so that the general construction is that indicated in the diagram, Fig. 213, which shews a generalised vertical section; *ABCDE* is the hutch, the upper portion of which is prismatic, the lower part being either prismatic, semicylindrical or pyramidal. The hutch is divided longitudinally into two compartments by the division *HI*, which only extends part-way down so as to leave an ample space for communication between the two compartments below the division *HI*. In the back compartment, which is sometimes the smaller, but also often of the same size as the other, works the piston *KL* attached to a piston rod which in turn is connected to suitable driving mechanism. The motion is usually produced by an eccentric, a double-disc eccentric, by means of which the length of throw can be varied, being often used; sometimes a crank and link are used, this being a well-known mechanical combination for giving a slow upward and quick downward motion. Several other methods of actuating the piston are used, but are rare. The front compartment contains the sieve *FG*. The tailings or lighter portions of the jigged material (waste in the case of ore, clean coal in the case of coal) escape over the weir or gate *ab*, the height of which above the sieve is usually adjustable, and in some cases the width also. The weir may be either at the side or at the front of the hutch, being always at the side opposite to that at which the material is fed on to the sieve. The sieve may exceptionally slope slightly from the feed side towards the discharge weir, but only when very coarse stuff is being jigged, and never in jigging through the sieve.

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[1] *Amer. Inst. Min. Eng.*, "Concentrating Magnetite with the Conkling Jig," by F. S. Ruttmann, Vol. XVI. 1888, p. 609.

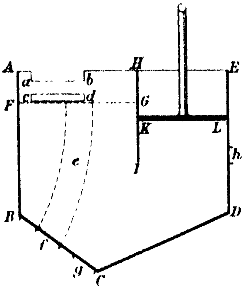

Fig. 213. Diagram of construction of jig with fixed sieve.

When this method of working is adopted, the concentrates accumulate in the hutch and a gate, as at *g*, is provided from which they can be drawn off intermittently or continuously. In jigging over the sieve there must also be a similar gate, as a certain amount of hutchwork always forms and has to be removed from time to time. With the latter system of jigging, the concentrates may either be discharged through a gate above the sieve, as at *cd*, the height and width of which are adjustable, or else a pipe is provided as indicated by the dotted lines *e*, which opens from the centre of the sieve, and through which the concentrates are drawn off continuously or intermittently. Several different forms of discharge are in use, some being arranged for taking off several different products at different levels above the sieve. It seems to make very little, if any difference, at what point on the sieve the concentrates are drawn off, as they move very freely over its surface, being in the state of quicksand or acting almost like a liquid. A water supply, sufficient to make up the loss of water discharged with the various products, has to be provided; a very usual position for it is indicated at *h*; it is sometimes provided with valves opening inwards into the hutch.

When these jigs are arranged as multiple-compartment jigs, each sieve is placed at a slightly lower level than that of the preceding one, and the discharge weir of each sieve becomes the feed shoot of the next, each compartment taking out one particular grade of concentrate, as shewn e.g. in Fig. 217.

Hutches may be made either of wood, sheet iron or cast iron. The former is light, cheap, easy to erect, but liable to decay; wrought iron is strong and durable except in treating ores containing easily decomposable sulphides. Cast iron makes a strong but very heavy hutch. Wooden hutches have either straight sides, or else the lower part is hemispherical, being built of barrel-like staves, held together by iron straps furnished with tightening bolts. Wrought iron hutches in the same way may have either straight sides or be rounded below. Cast iron hutches are usually pyramidal. The pyramidal shape is especially convenient in jigging through the bed, as the hutchwork is readily drawn off from the apex of the pyramid. Sieves are made either of wire gauze or of punched plates, the former being best suited for the finer, the latter for the coarser sizes. Brass wire may be used for the finest sizes, otherwise steel wire is preferable except for ores that carry easily decomposable sulphides, which when standing would yield sufficient sulphuric acid to corrode the iron; under the same circumstances, sheet copper may be used instead of steel plates. The sieves must be secured to a wooden or iron frame; to prevent their being buckled or burst by the weight upon them they must be supported by a grid of wooden or iron bars; these bars are best V-shaped in cross-section, so as to oppose the minimum of resistance to the flow of the water Very fine wire sieves may also with advantage be supported upon a strong sieve of coarser mesh. The size of the sieves varies from 2 to 3½ feet by 1½ to 2½ feet for ore, up to about 5 feet by 3 feet for coal; they are usually placed 1 foot to 1½ feet below the top of the hutch. The upper portion of the hutch above the sieve should be lined with boards ½ inch to 1 inch thick to prevent undue wear of the sides by the moving mass of mineral. The piston is usually built up of planks of hard wood such as oak or elm, bolted together and strapped with iron above and below, the piston rod going through it and being secured by a nut or a cotter. Long pistons, say over 3 feet in length, should have two rods. The piston should be hung at about the same height as the sieve or a little lower at half stroke; it should never rise clear of the water in the hutch, so as to avoid violent shocks. The piston is practically never tight fitting, but works with about ⅛ inch to

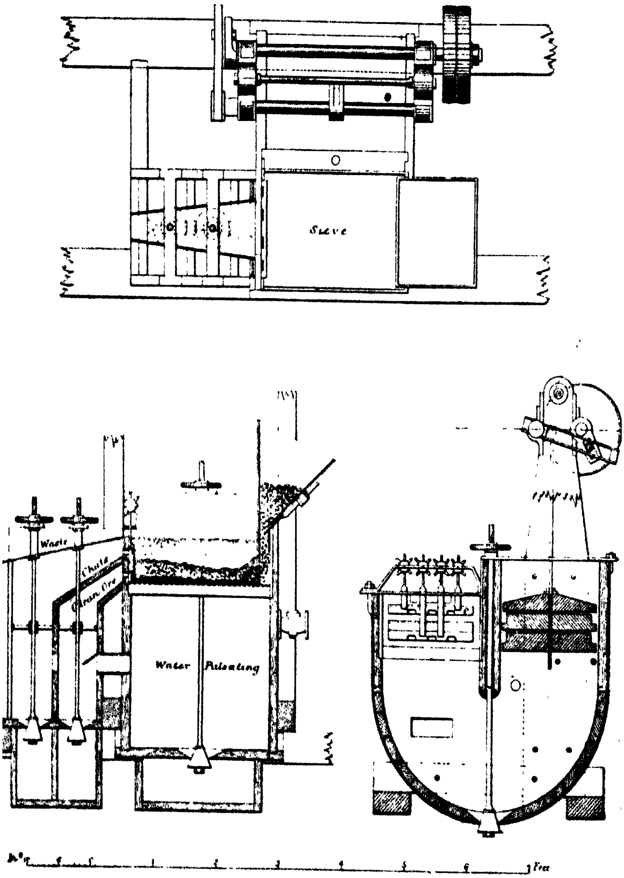

Fig. 214. Single compartment jig. Plan, longitudinal and transverse sections.

¼ inch clearance all round; the compartment in which it works should be lined with boards that can easily be renewed. The division between the piston and the sieve compartments should be of such depth as to equalise the ascending currents of water over the area of the sieve; its depth is usually 7 inches to 12 inches, being deeper the wider the sieve. The area of the piston is usually equal to that of the sieve, and is best made so; in a few special constructions the piston area is only half the sieve area, but this plan is not to be recommended for ordinary Harz sieves.

A typical single compartment jig[1], jigging over the sieve and making three products, clean ore, middlings (called "chats" in the North of England, "raggings" in Cornwall) and waste (known as "cuttings" in the North of England) is shewn in Fig. 214. This form is rather out of date but illustrates well the principle of discharging two grades of products through weirs above the sieve. This machine is worked with a quick downward and slow upward stroke of the piston, by means of the mechanism already referred to.

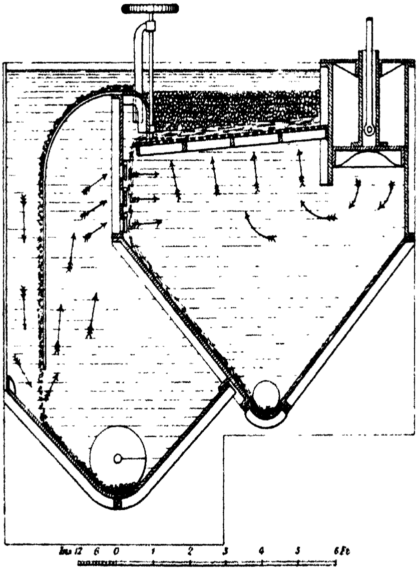

Fig. 215. Sheppard nut coal washer. Vertical section.

Fig. 215 shews a jig of the same type as built by Messrs Sheppard and Sons, Ltd. for washing coarse coal; nut and pea coals, screened between 1½ inches and ⅜ inch mesh, are washed on such a jig, duff coal,

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[1] *Inst. C. E.*, "The Dressing of Lead Ores," by T. Sopwith, Vol. xxx. 1870, p. 106.

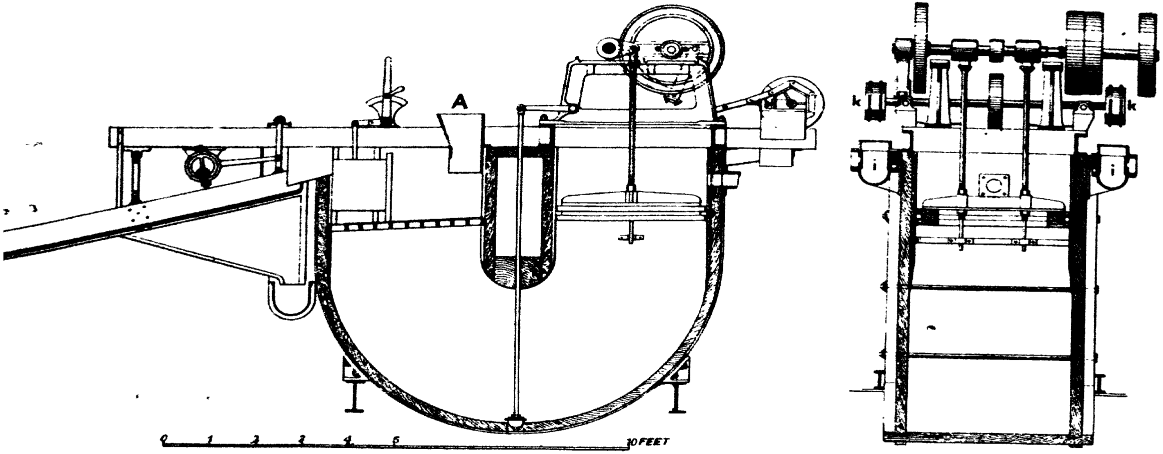

Fig. 216. Skoda nut coal washer. Longitudinal and transverse sections.

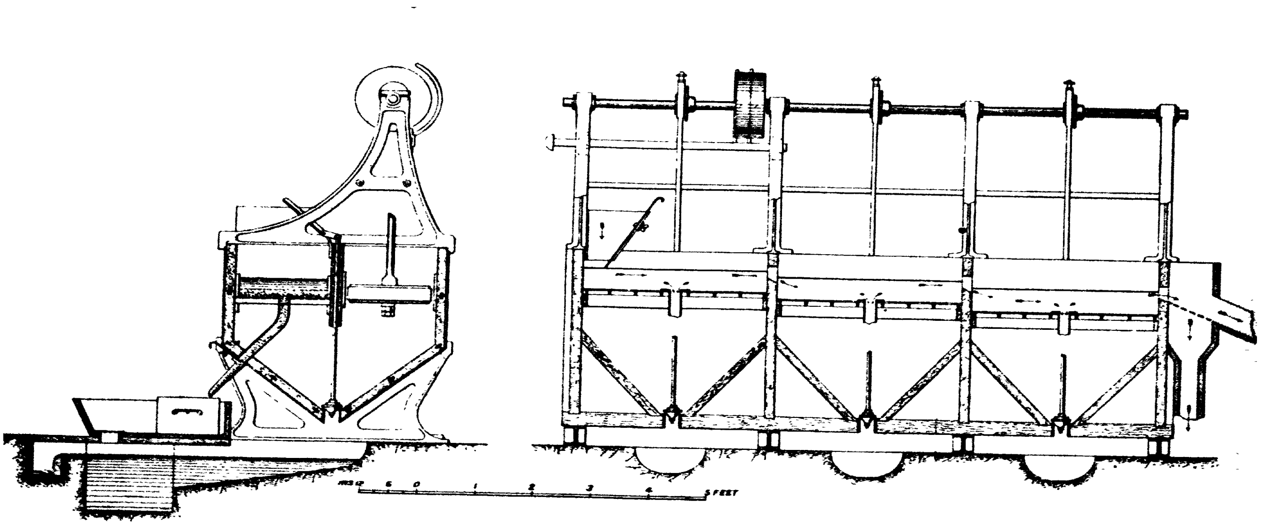

Fig. 217. Central discharge jig. Longitudinal and transverse sections.

screened through $\frac{3}{8}$ inch mesh, being washed on felspar washers. The ji is built of cast iron, the piston also being of iron; it will be notice that the stroke of the piston is guided so as to be truly vertical, the connecting rod being as shewn free to swing through a sufficient angle t admit of this. The piston can therefore be made tight fitting. The siev is set on a sufficient slope to help the coals to travel over it. The dir is delivered through an adjustable gate, which is so arranged as to allo it to drop into the main hutch, whence it is discharged together wit anything that may come through the sieve, by means of a screw conveyor. The clean coal drops over a weir into an outer case from which it is also removed by a screw conveyor, whilst the water flowing over the weir is drawn into the hutch again by means of valves situated in the front wall of the hutch. The piston makes 60 to 70 strokes, 5 inches to 7 inches long, per minute, and at this rate each hutch can treat about 5 tons of coal per hour.

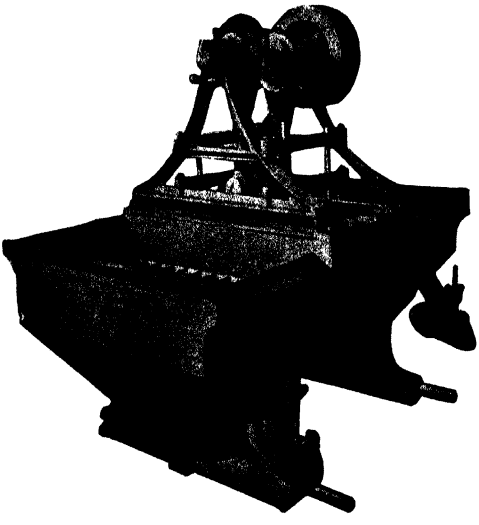

Fig. 218. Central discharge jig. Perspective.

Another form of nut-washer, the **Skoda** machine, is shewn in Fig. 216[1]. In this the coal to be washed enters through the small hopper shewn at *A*; the dirt flows out through adjustable gates on either side into troughs, *i*, from which it is removed by chain scrapers, *k*. The coal is discharged over the edge opposite to that where the hopper is placed, and drops on to a small jigging screen, which delivers it uniformly at the same time draining off the bulk of the water, which is returned to the jig.

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[1] *Kohlenaufbereitung*, Lamprecht, Pl. xxi.

A three-compartment jig[1] for jigging over the sieve is shewn in Fig. 217, the sieves being furnished with central tube discharge. There is also a plug to enable any hutchwork to be drawn off. A perspective view of a pair of jigs on this principle with cast iron hutches as made by the Gates Iron Works is shewn in Fig. 218.

A typical three-compartment jig, as made by Messrs Fraser and Chalmers, Ltd., for jigging through the sieve is shewn in Fig. 219, a perspective view of a similar machine being shewn in Fig. 220. This is a form of wooden jig that is very widely used. Another form that is also in great favour being shewn in Fig. 221, which shews details of its construction. A machine upon quite the same principle but having a semicircular hutch-bottom is shewn in Fig. 222 as made by the Grusonwerk Co. of Magdeburg. The construction, when iron plate is substituted for wood, is represented by a two-compartment jig, Fig. 223. In the former style, the makers quote about £50, £70 and £90 for one-, two- and three-compartment jigs respectively, the sieves being about 32 by 16 inches.

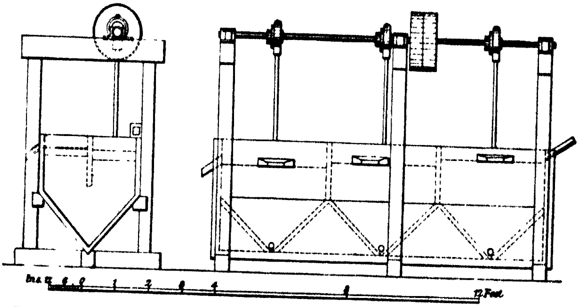

Fig. 219. Three-compartment jig with wooden hutch. End and side elevations.

It will be seen that all the above jigs differ only in the less important details of construction, and that by combining in different

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[1] *Inst. C. E.*, "The Concentration and Sizing of Crushed Minerals," by R. S. Commans, Vol. CXVI. 1894, p. 9.

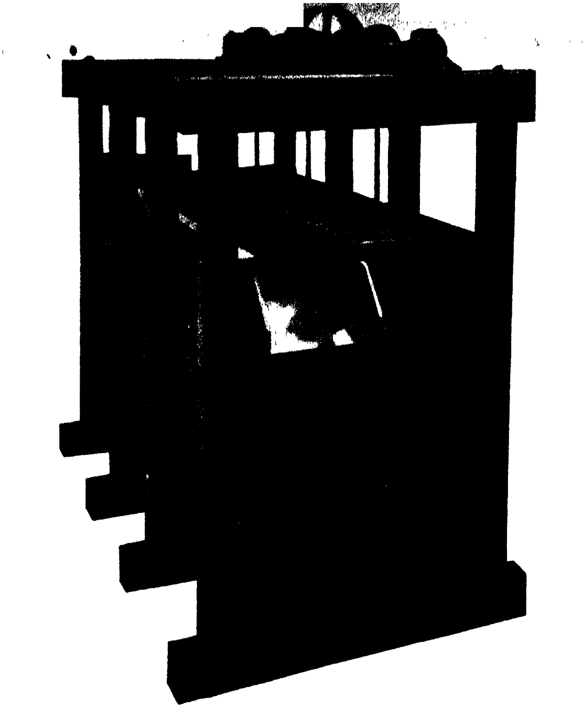
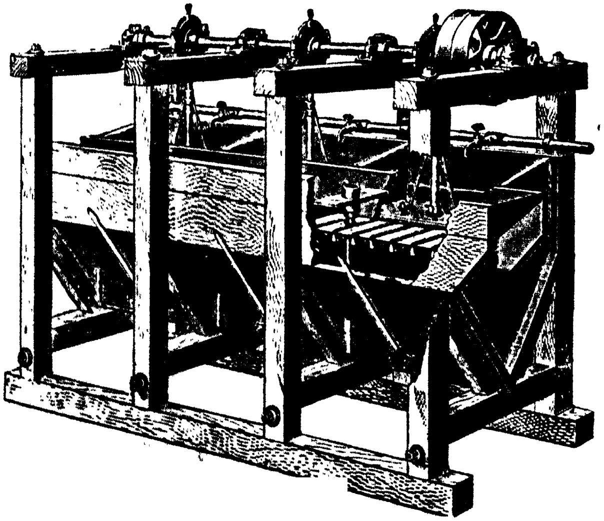


Fig. 220. Three-compartment jig with wooden hutch. Perspective



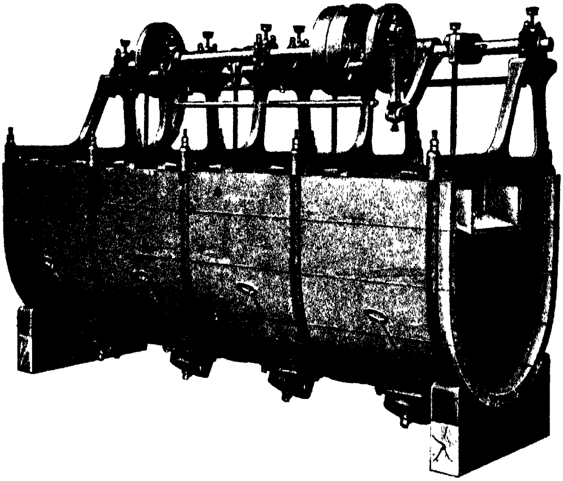


Fig. 222. Four-compartment jig with wooden hutch. Perspective.

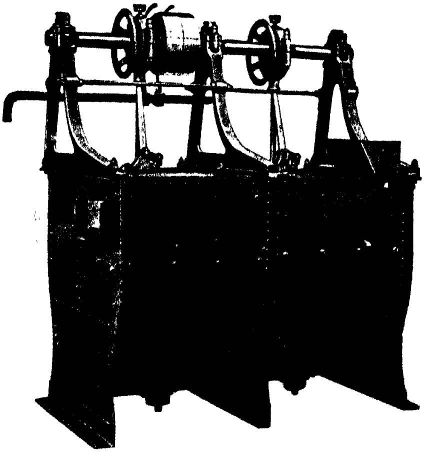


Fig. 223. Two-compartment jig with iron hutch. Perspective.

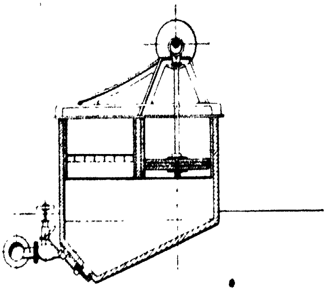
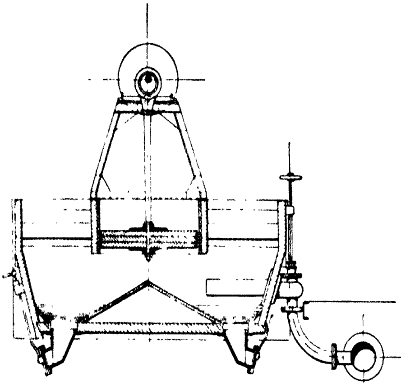
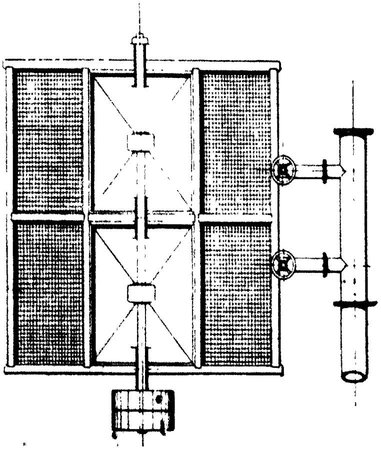


Fig. 224. Single Felspar washer. Vertical section.





ways the various arrangements shewn, a large number of modifications would result.

The jig shewn in Fig. 224 is a **Felspar Washer** as used for small coals, made by Messrs Schüchtermann and Kremer, Fig. 225 shewing

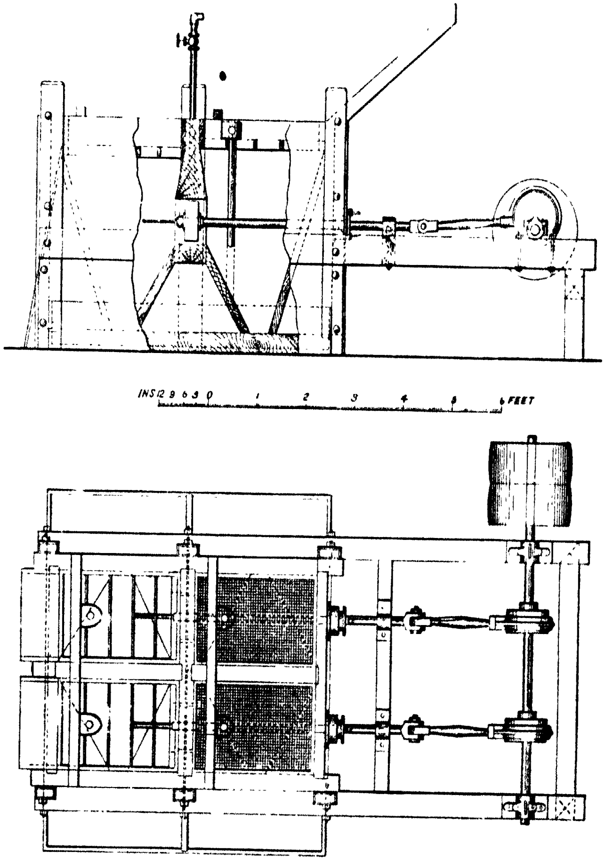


Fig. 226. Jigs with horizontal piston. Sectional elevation and plan.

a double washer of the same type by the same makers. The principles underlying this form of washer have already been discussed (p. 264), and a Sheppard felspar washer with movable sieve has been shewn in

Fig. 209; it is however more usual to build felspar washers with fixed than with movable sieves, and the ordinary felspar washer is often built exactly like a two- or three-compartment Harz jig shewn in Figs. 219 to 223.

In Fig. 226 a set of four jigs is shewn, the pistons of which are arranged horizontally instead of vertically, one piston being thus sufficient for a pair of sieves. This arrangement is compact and comparatively cheap, but suffers from the disadvantage that each sieve of the pair must be worked at the same piston speed and stroke; moreover the piston is apt to be injured by fine mineral matter suspended in the hutch water, and is difficult of access for repairs; there is also a stuffing-box in the side of the hatch to keep tight. It will be noticed that the two compartments are alternately in pulsion and in suction, whereas in the ordinary Harz jig pulsion and suction may take place either simultaneously, alternately, or at any other interval of time, as may be desired. The piston is circular and is packed watertight, working in an iron cylinder. Such jigs are used quite exceptionally in Missouri and in North Wales for dressing lead ores. Their use is not extending.

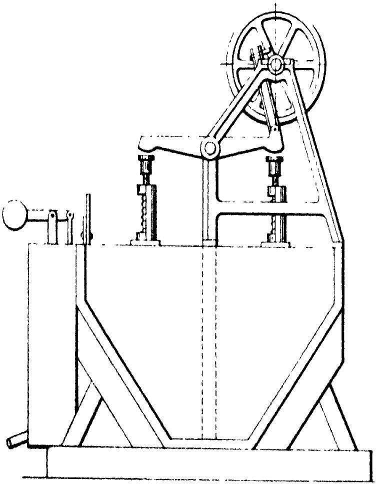


Fig. 227. Collom jigs. Side elevation.

The jig known as the **Collom** jig is shewn in Figs. 227, 227*, and 228, the former shewing a set of two jigs in a wooden frame, whilst the latter

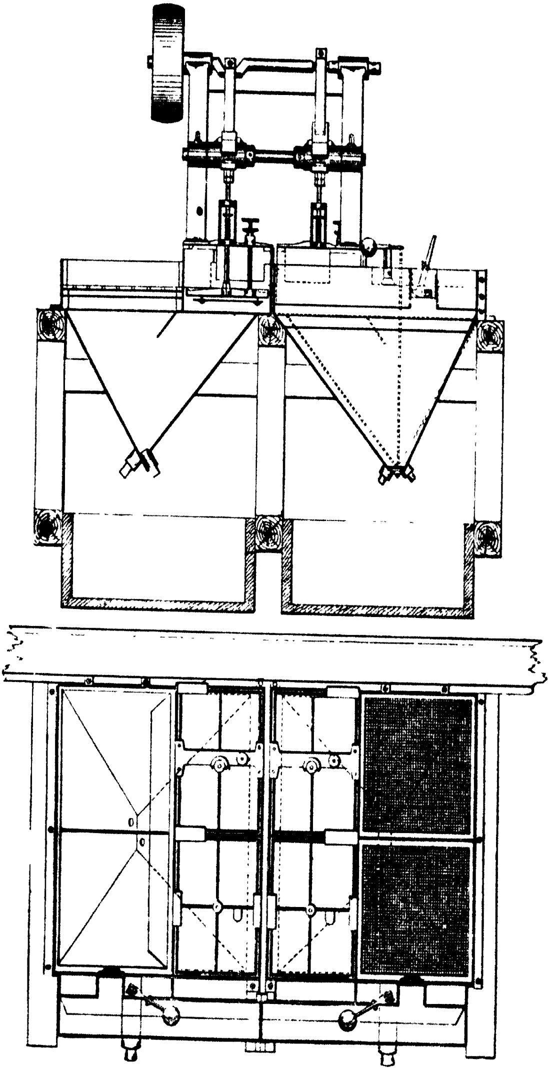


Fig. 227a. Collom jigs. Plan, and front elevation.

shews them on a light iron frame; these shew independent jigs built in pairs in one hutch with a piston compartment between them. The latter is divided completely into two separate halves, front and back, one of which communicates with the space below the right hand, the other with the space below the left hand sieve. The pistons are attached to short vertical rods that terminate in heads with indiarubber buffers, below which are coiled powerful spiral springs, the compression of which can be adjusted as desired. A T-shaped rocking piece is so placed above the heads of the piston rods that each of them is depressed by it alternately, thus producing the downstroke of the piston, whilst the spring lifts the piston again, thus making the upstroke. The piston has thus a rapid downstroke and a slower upstroke. It will be noted that the piston area of these jigs is less in proportion than in the ordinary Harz jig, being only about half the area of the sieves.

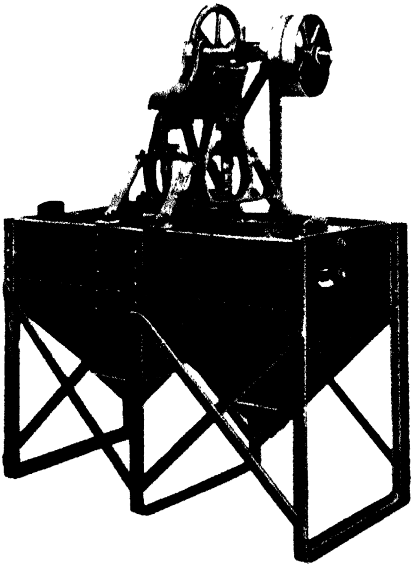


Fig. 228. Collom jigs. Perspective.

Collom jigs are only in use in a few places, chiefly on Lake Superior for treating ores of native copper, also in Colorado, at Broken Hill, N.S.W., and a few other places. It does not appear that they possess any advantage over the Harz jigs, and the wear and tear seems to be greater.

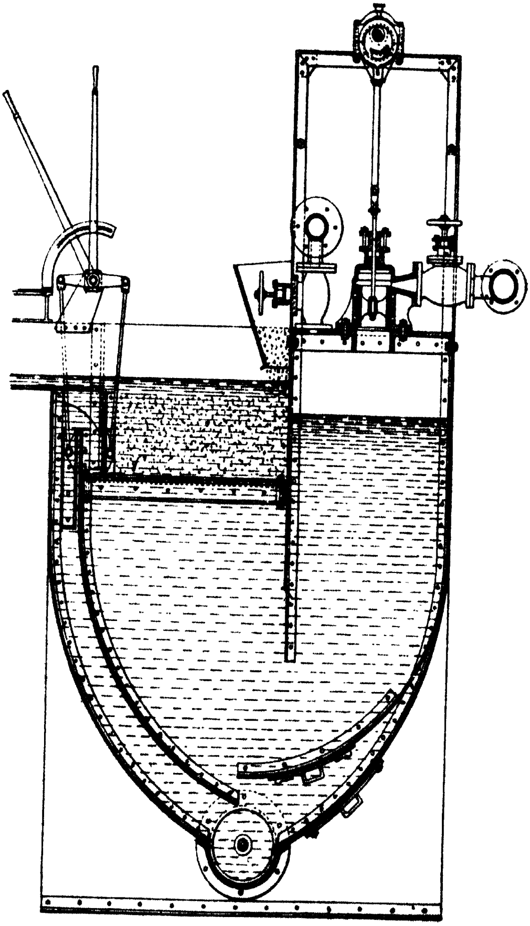


Fig. 229. Baum jig. Vertical section.

Mr Arthur Taylor[1] has recently proposed working jigs by a vibromotor instead of by an eccentric; he seems to prefer to cause the entire hutch to move, making two sides of the hutch of some flexible material, *so that pulsation of water through the sieve is produced by the motion of the hutch and attached sieve*, the inertia of the water contained in the hutch causing the latter to move differentially as compared to the former. He also claims the use of a swinging sieve actuated by a vibromotor. In either case he proposes to work at from 500 to 1500 strokes per minute, and states that he can jig successfully "the finest slimes that can be treated by any mechanical means." This system has not yet undergone any actual practical tests.

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[1] *Bull. Inst. Min. Met.* October 1908.

The **Baum Jig**[1] is used for coal washing. It is shewn in section in Fig. 229. Here the pulsation of the water is produced not by the motion of a piston, but by puffs of compressed air acting upon the surface of the water in a compartment by the side of that carrying the screen. Air at a pressure of 1½ to 2 lbs. per sq. inch is admitted in puffs by means of a valve driven by a small eccentric. In the machine shewn in the figure, jigging takes place over the screen, the dirt passing through a gate into the hutch whence it is removed by a screw conveyor. For coarse coal from 2 inches to 7/16 inch, the valve makes 50 to 70 strokes and for fine coal below 7/16 inch, from 75 to 110 strokes per minute. Baum keeps a deep layer of coal on his sieve and maintains that with his jig a felspar bed is not required, the shale in the coal forming a sufficient bed.

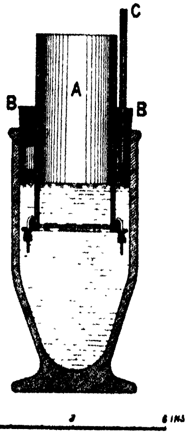

Fig. 230. Braun laboratory hand jig. Vertical section.

A convenient little jig for laboratory tests made by Braun of Freiberg and worked by puffs of air may fitly be included here. As will be seen from the illustration it consists of an outer vessel of glass about 4 inches in diameter and 7 inches high. Into it fits an indiarubber ring *BB*, through which passes a glass cylinder *A* open at both ends, to the lower end of which sieves about 2¼ inches in diameter, and of varying mesh as may be required, can be clamped. Through the indiarubber ring passes a brass tube *C*, which terminates below in an annular tube with numerous small holes whilst its upper end is connected by a piece of indiarubber tube, with a hollow indiarubber pear-shaped bulb some 4 inches in diameter, by alternately pressing and releasing which, jets of air are made to issue into the upper portion of the glass vessel through the perforations in the annular portion of the brass tube; the variations in pressure thus produced, cause the water poured into the outer vessel to pulsate through the sieve. A small sample—say ¼ lb. —of crushed mineral may conveniently be jigged in this little apparatus and the conditions for successful working thus determined. In the writer's laboratory this little instrument is always used for jigging through the sieve, a bed of leaden shot being employed. It is a useful little appliance and can be recommended for jigging tests.

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[1] *Trans. Fed. Inst. Min. Eng.* VII. p. 159.

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CHAPTER VIII.

HORIZONTAL CURRENT SEPARATORS.

THE action of a horizontal current of water in separating particles of different ultimate falling velocities has already been explained; it has been shewn that such particles falling in a horizontal or practically horizontal stream of water will reach the bottom of the stream at distances from the point of their introduction which will be greater in proportion as their ultimate falling velocities are less. The separation thus takes place entirely whilst the particles are falling through the body of the stream of water. It has however been also shewn that a separation can be effected between particles of different specific gravities if these are subjected to the action of a shallow stream of water, the friction of the particles against the solid surface over which the stream flows playing an important part in this separation. It is obvious that this latter condition necessarily prevails in the lower portion of every stream of water, so that it is quite possible for separating action of the latter kind to take place in addition to that due to the former principle. Accordingly we have to consider appliances acting by separation in a deep stream of water, those acting by separation in shallow streams, and an intermediate class in which both modes of action take place. Although the two actions are quite different in kind, yet they may take place to varying extents in appliances differing only in degree, that is to say, in depth of current, so that there is in practice no sharp dividing line between the two sets of appliances, although we can distinguish easily between appliances at either end of the scale. Those working on the first-named principle alone are especially suitable for the treatment of larger particles and coarse sands, whilst those working purely on the last-named are especially adapted for fine sands and slimes; those that combine both principles are not suitable for treating the last-named class.

Appliances making use of a deep stream of water are usually employed only to effect a preliminary separation and have then to be followed up by more accurately working appliances, but may at times be used for finishing operations, especially when the bodies to be separated have widely different specific gravities.

The typical appliance to be considered under this head is the

Box Buddle or rectangular buddle, a very usual form of which is shewn in Fig. 231. It consists of a rectangular wooden box 10 to 15 feet long, 4 to 6 feet wide and 1 foot 6 inches to 2 feet 6 inches deep, set at a gradient, ranging from 4° to 10°, the inclination being greater the coarser

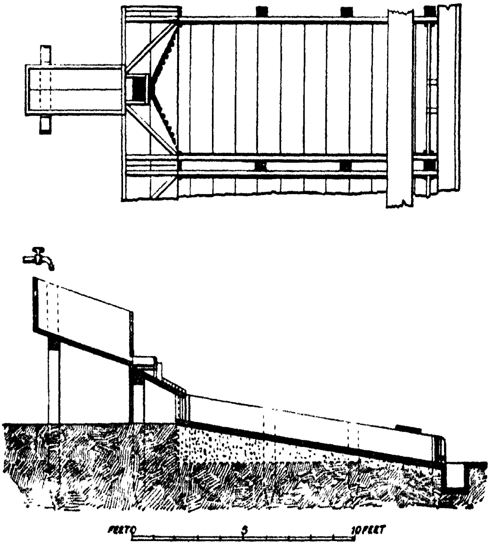

Fig. 231. Box buddle. Plan and sectional elevation.

the material to be treated. At the upper end there is a headboard set at a slope of 15° to 20°, of the full width of the buddle and 15 inches to 20 inches broad; it is furnished with a row of "pins," these being diamond-shaped blocks of hard wood, about 2 inches by 1 inch, which can turn on a central pivot, thus varying the width of the passage between adjoining pins and providing a simple means of regulating the inflowing stream of pulp. By properly adjusting these pins it is easy to arrange that the stream flowing over the headboard shall be of uniform depth right across the buddle. Above the headboard is a box, triangular in cross-section, into which the material to be treated is shovelled by hand, whilst a stream of water plays upon it; the labourer who shovels in the mineral keeps it stirred up with his shovel, although in some buddles a stirrer consisting of a short horizontal shaft studded with teeth and driven mechanically is employed. The pulp flows into a sieve that arrests all lumps of stone, chips of wood, etc. that may have accidentally found their way into the crushed mineral. The tailboard of the buddle is pierced with two vertical rows of holes about 1 to 2 inches in diameter, fitted with plugs, and set at distances apart a little less than the diameters of the holes, the holes in the two rows alternating with each other. By successively plugging these holes the level of discharge from the lower end of the buddle can thus be gradually raised. Across the lower part of the buddle lies a plank upon which stands a labourer armed with a long handled besom with which he continually sweeps up and levels the depositing mass of mineral, so as to keep it smooth and to prevent channels from forming in it. Two men are needed to each buddle; one shovels in the mineral and

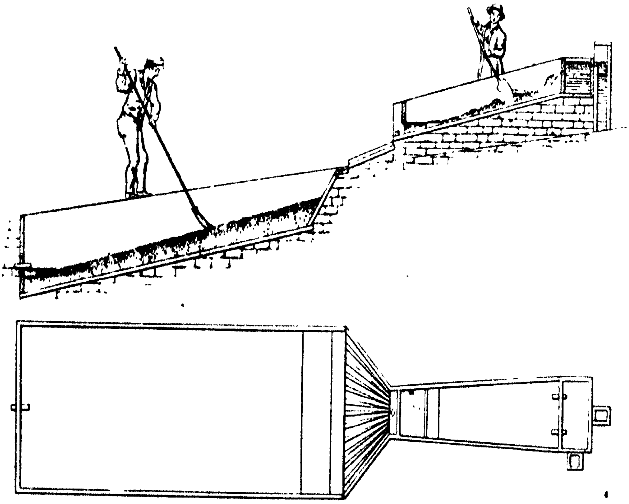

Fig. 232. Box buddle. Plan and sectional elevation.

regulates the water supply and the pins on the headboard; the other uses the besom and puts the plugs into the tailboard as required. Another somewhat similar form of buddle is shewn in Fig. 232[1], which also indicates how the appliance is worked. The pulp flows over the headboard into the buddle, and in virtue of the principles already explained, the heavier particles are deposited close to the headboard, and the lighter ones further away from it along the buddle; the plugs are put in at the bottom so as always to keep a little sump of water at the bottom end. The sweeper sweeps always from below upwards, drawing his broom slowly from side to side. The buddle is thus gradually filled, the operation being completed when it is about one-half or two-thirds full at the headboard end, and the plugs are then pulled out to allow the mass of mineral to drain. The mass is then divided into three sections (four are made exceptionally) by the foreman, who usually marks them out with a shovel. The top section or "heads" is clean enough for the next process, the middle section has to be buddled over again, whilst the lowest section consists of barren tailings and can be thrown away. The heads usually require further treatment, or they may only need "tossing" or "chimming" to be ready for the market. Such a buddle takes about 1 cubic foot of pulp per minute, containing about 28 lbs. of dry sands; two men will work off four buddles, each holding about 30 cwt. of mineral, in a 10 hour shift. Usually such buddles are worked in pairs, two men buddling in one, whilst two others empty the other, and so on alternately. This appliance is simple in the extreme and inexpensive to construct and has no moving parts, but the labour cost is very high. It is accordingly practically obsolete, having been replaced almost everywhere by the round buddle.

The **Round Buddle** works on the same principle as the above; its construction may be compared to that of a series of box buddles set radially so as to form a circular box; if the hypothetical box buddles had their headboards at the centre, the convex buddle would result, if at the circumference, the concave buddle, both forms being in use, though the former is the more extensively employed.

The **Convex Buddle** is shewn in Fig. 233[2]. It consists essentially of a wooden floor in the shape of a very obtuse cone, the angle to the horizontal being between 4° and 8°. It is usually built in a shallow circular pit about 18 inches deep, but is sometimes raised above the surface of the ground. Its diameter ranges usually from 18 to 24 feet;

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[1] *Proc. Inst. C. E.* 1858, Vol. XVI. Pl. 6.

[2] *Proc. Inst. Mech. Eng.* 1873, Pl. 41.

in exceptional cases buddles up to 50 feet in diameter have been built in Cornwall. In the middle stands a circular headboard 6 to 10 feet in diameter, sloping at a steeper pitch than the floor of the buddle; the lower edge of the headboard being about 12 or 18 inches higher than

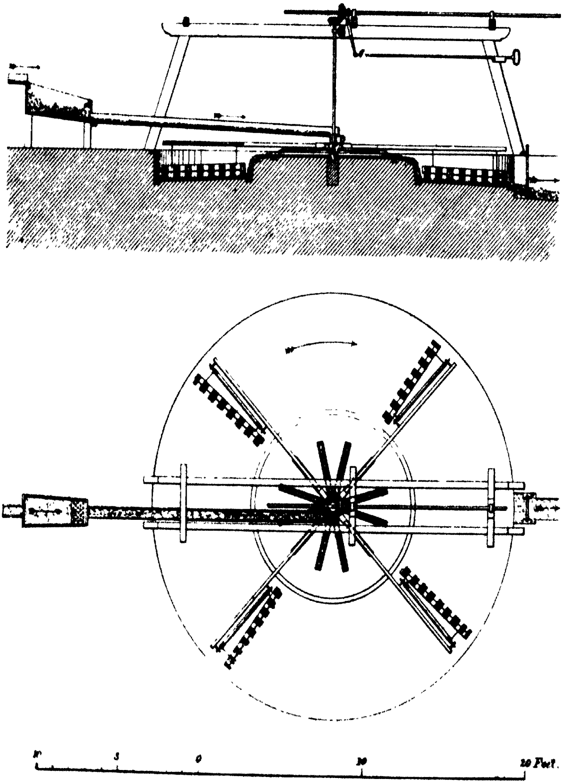

Fig. 233. Round convex buddle. Plan and sectional elevation.

the conical floor. The headboard may be furnished with regulating pins. The pulp to be treated is conveyed to the centre of the headboard and is discharged upon it in various ways. In the buddle shewn it runs into a central cup, from which it flows through a number of spouts which are slowly rotated so as to produce a uniform stream; in this case no regulating pins are required. In another form the pulp flows into a central funnel, the lower portion of which is partly closed by an iron cone so as to leave an annular space about 1 inch wide for the discharge of the pulp. In the centre is a vertical shaft driven by bevel gearing, to which are attached from two to six arms, which in their turn carry brushes to keep the surface of the deposited mineral smooth. These brushes are either long narrow besoms, or are simply strips of stout canvas nailed to a narrow board. These are suspended from cords wound round rollers, so that they can gradually be raised as the level of the accumulating material rises. As the deposit is greatest close to the headboard, these brushes are occasionally made in two lengths, each capable of independent adjustment. This adjustment is usually performed by hand, though in some exceptional cases the rollers are geared to the central spindle and driven by it. The arms make from 5 to 10 revolutions per minute. Such a buddle can take from 1½ to 3 cubic feet of pulp per minute, carrying from 28 to 56 lbs. of dry sands per cubic foot of pulp; there is, however, great variation in the rate of working of buddles, the filling of a buddle taking from 3 to 10 hours according to circumstances, and the buddle holding from 6 to 12 tons of mineral. The excess of water flows off either all round the buddle or through a trough at one side, the height of the overflow being adjustable as in the rectangular buddle. In either case the water should pass to settling pits to allow any fine mineral in suspension to be deposited. When the

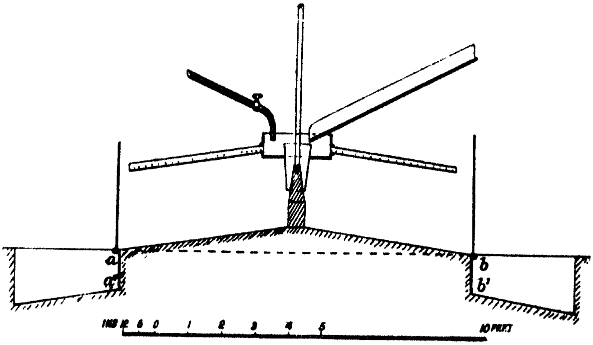

Fig. 234. Ring buddle. Vertical section.

buddle is full, the contents are divided into three parts by concentric circles, the heads going generally to the concave buddle for further cleaning, the middlings being buddled over again, and the tails being looked upon as barren waste. Buddles are usually built in sets; one man can look after some half-dozen of such ordinary convex buddles. The power consumption is very low, say about 0·1 H.P., and the cost of the appliance is but small; it can usually be built by an ordinary mine carpenter for an outlay of about £25.

A small buddle, shewn in section in Fig. 234, of somewhat similar construction, is in use in the Weardale district for treating lead ore, and is known as the **Ring Buddle**; the conical wooden floor is surrounded by a gutter about 12 inches deep, in which rests a ring of stout sheet iron, *aa'*, *bb'*, of diameter equal to that of the buddle. This ring is suspended by rods so connected with the driving gear that as the buddle fills the ring slowly rises and thus raises the level of the overflow. It will be seen that this buddle has no headboard, the pulp flowing through a conical distributor with an annular aperture surrounding a central post; instead of brushes to keep the surface of the deposit smooth, 4 arms of iron pipes, pierced with fine holes, through which issue jets of water, rotate slowly, and not only perform the office of the brushes, but also wash away some of the barren tailings. Both of these latter devices are also in use in the lead-mining districts of the North of England for buddles working without the ring. It seems to be very generally held that the headboard is of doubtful utility and may well enough be dispensed with, but the use of jets of water instead of brushes is apt to give rise to the formation of channels in the deposit, in which valuable mineral may be washed away.

The **Concave Buddle** is shewn in Fig. 235[1]. As will be seen, the floor is funnel shaped, the slope of the cone being again very flat. The pulp is conveyed as before to the centre of the buddle, but from the central funnel long spouts convey it to a narrow peripheral headboard. The central shaft carries arms and brushes just as in the convex form; sometimes these brushes are attached to the pulp spouts, which then act in a double capacity. In this form of buddle the heads are deposited at the circumference and the tails nearest to the centre, so that there is more space available for the former than for the latter; it is therefore well adapted for cleaning partially concentrated material such as the heads from the convex buddle, and is generally used for this purpose.

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[1] *Proc. Inst. Mech. Eng.* 1873, Pl. 43.

Borlase's Buddle (Fig. 236[1]) is a concave buddle with a sliding ring worked by hand, by means of which the level of the overflow can be very accurately adjusted. The ring is shewn on a larger scale at *RR*, and it is lifted, as shewn, by a lever and screw gear, which works a spindle that passes through the shaft of the buddle.

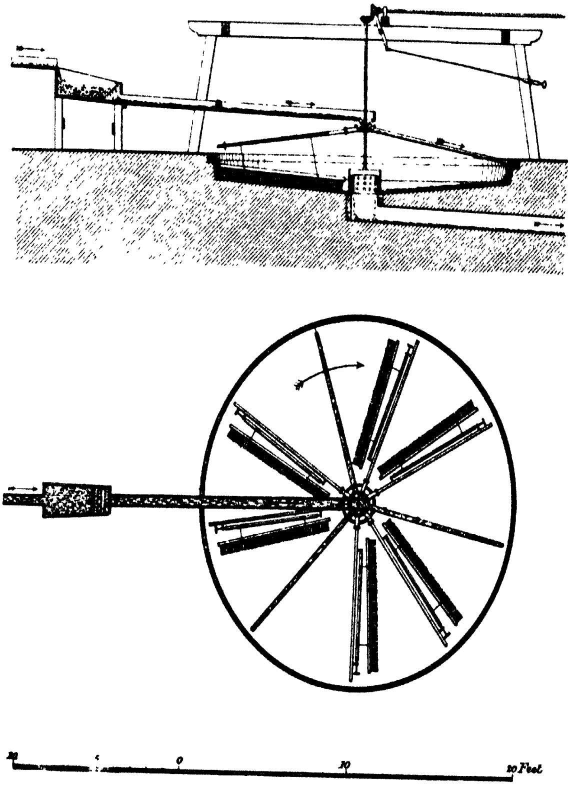

Fig. 235. Concave buddle. Plan and sectional elevation.

Appliances that depend partly on separation by falling through a horizontal current of water and partly upon the differential rate of travel along the bottom of the stream have been in use from im-

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[1] *Proc. Inst. Mech. Eng.* 1857, Pl. 44.

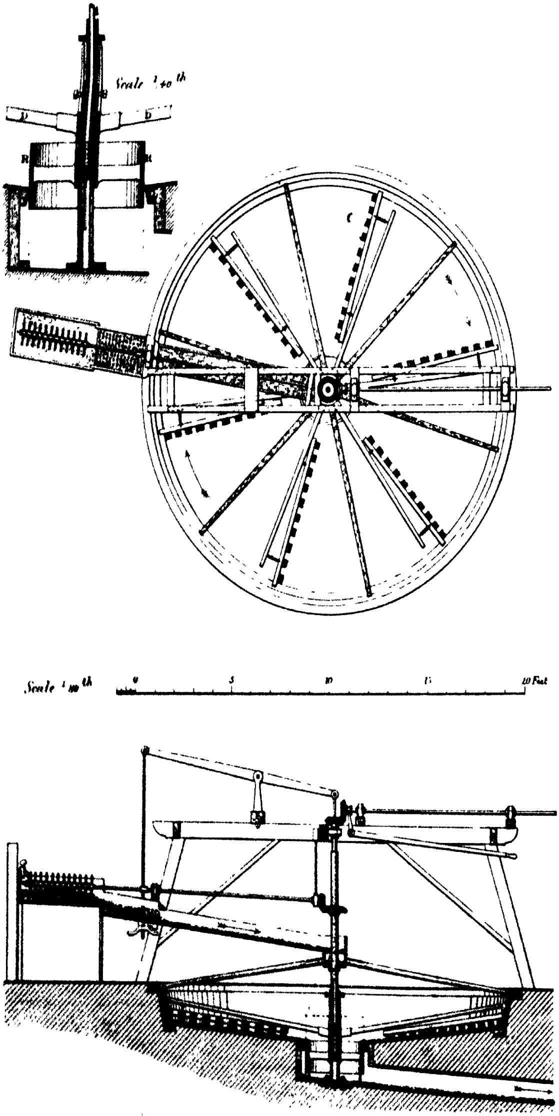

Fig. 236. Borlase's buddle. Plan and sectional elevation.

memorial periods, and constitute the most primitive type of dressing plant. Illustrations of these are to be found in Agricola's well-known work, and they are to be met with in closely related, but without doubt independently invented forms, amongst semi-civilised races all over the world. As usually constructed they consist of a wooden trough at the head of which the pulp to be treated is fed in, an additional stream of clear water being sometimes run in nearer to the head of the trough. The inclination of the trough, the quantity of water, and the resulting velocity of the current are usually so proportioned to the size and weight of the particles of mineral that the heavier mineral remains at rest in the trough, whilst the lighter is carried off. In order to prevent particles of the lighter material from being entangled in and held back by the heavier, it is necessary to work the material lying on the floor of the trough up against the current, which is usually done by means of a besom, a fine-toothed rake, or a hoe. The clean heads thus obtained are usually shovelled or raked out from time to time, whilst the waste runs off continuously. Such an arrangement is used, for example, by the Chinese in the Malay Peninsula for concentrating the tinstone from the tin-bearing alluvial deposits.

The "**Strips**," which were largely used, and are still employed to some extent for a preliminary concentration of crushed tinstuff in Cornwall, consist of troughs 20 to 30 feet long, 12 inches deep, and 18 inches wide, in which the material richest in tin is gradually deposited at the upper end, whilst the tails are practically barren.

Strakes and **Tyes** are very similar to the last named, but are usually shorter, namely, from 10 to 20 feet in length, and are usually worked with a hoe, whereas strips are not thus worked. The lower end of the tye is often arranged to be closed by a series of bars of wood about 1 inch deep, which are laid in successively as the material accumulates on the bottom of the trough, until the latter is sufficiently filled. Such appliances are also sometimes fitted with riffles; these in their simplest form consist of ridges of wood or iron ½ inch to 2 inches deep, placed across the trough to arrest the heavier particles; grooves in the bottom may take the place of ridges, and many special forms are used in hydraulic gold mining. These appliances, like rectangular buddles, are usually worked in pairs; they are not very efficient and have little but their cheapness to recommend them.

Slime Pits and **Labyrinths** work upon a somewhat similar principle; they are simply rectangular pits into which water carrying fine particles of mineral is run, and in which the velocity of the current

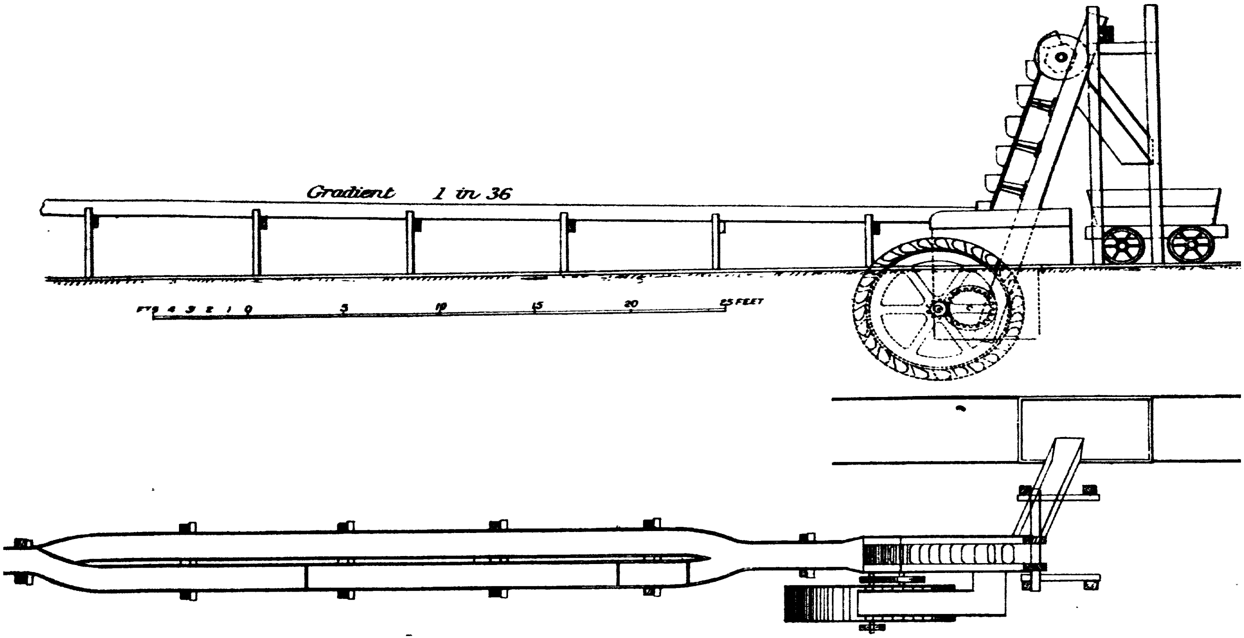

Fig. 237. Trough washer. Plan and elevation.

is sufficiently reduced to allow the fine particles to settle. The water is run off and the pits emptied out from time to time.

The ordinary **Trough Washer**, used for washing moderately small coal, say from 1½ inch to ¼ inch cube, works on exactly the same principle. In its simplest form it consists of a trough 16 to 24 inches wide and 9 to 18 inches deep, through which a stream of water carries the dirty coal; riffles are set across the trough at intervals of 10 to 20 feet. The stones and dirt sink to the bottom and remain there, being caught and retained by the riffles, whilst the lighter clean coal flows over them and runs off at the lower end of the trough. These troughs are worked in pairs, and the operation must be stopped before the dirt reaches the level of the upper edge of the riffles, when the stream of coal and water is diverted into the neighbouring trough, and the accumulated dirt shovelled or scraped out.

Such a washer is shewn in Fig. 237 as used at Flimby Colliery, near Maryport, Cumberland[1]. There are two iron troughs 150 feet by 17 inches wide and 13 inches deep, set at a gradient of 1 in 36; each has two riffles, about 2 inches deep, one at the lower end and one some 20 feet higher up. The troughs are used alternately, the stream being diverted from one to the other by means of a movable tongue placed at the upper end. The velocity of the stream is 300 feet per minute, and the appliance treats 10 tons of coal per hour with a water consumption of 400 gallons per minute. About 18 per cent. of dirt is taken out of the coal, and the working cost of washing is given as 1·18*d.* per ton of coal treated.

At Tredegar[2] there are three troughs, side by side, 250 feet long, 32 inches wide at the top and 18 inches at the bottom, by 12 inches deep, set at a gradient of 1 in 30; they are built of 2 inch plank, the bottom being covered with ¼-inch iron plates. Riffles, consisting of strips of iron, are inserted every 12 feet; these are 3½ inches deep at the head of the trough and get gradually shallower as they approach the lower end where they are only 1½ inches deep. The shale is removed from each trough in succession at intervals of about ¾ hour, the supply of coal being cut off from the trough that is being cleaned. This plant treats 20 tons of small coal per hour, the coal having been passed through a 1 inch screen.

An extremely primitive type of trough washer is in use in Asturias[3];

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[1] *Trans. Min. Inst. Scotland*, Vol. XI. p. 184.

[2] *Min. Proc. Inst. C. E.*, "Coal Washing," by J. F. Harvey, Vol. LXX. 1881, p. 106.

[3] *Trans. Inst. Min. Eng.* Vol. XXVIII. p. 423.

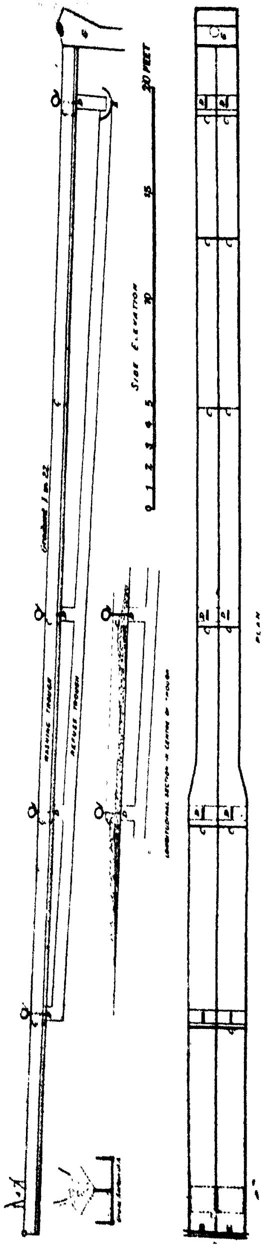

Fig. 238. Trough washer. Plan and sectional elevation.

it simply consists of a wooden trough about 20 feet long, 15 inches wide, and 15 inches deep set at a considerable incline. A stream of water runs through the trough, and the coal is shovelled in at the top end and worked about with the shovel; the dirt is raked out at the head of the trough, whilst the washed coal escapes at the lower end.

An improvement in the ordinary trough washer consists in having more numerous riffles, and removable slides in the bottom of the trough, through which the accumulated dirt may be removed. This form is well illustrated by a trough washer at Lodge Colliery, Slamannan[1], shewn in Fig. 238. This is a double trough 100 feet long, each trough being 2 feet wide at the upper end and 1 foot 6 inches wide at the lower. The bottom is of steel $\frac{1}{4}$-inch thick and the sides of $\frac{3}{16}$-inch iron. It is set at a gradient of 1 in 22. The coal is tipped into one or other of the troughs at *A*, a movable hopper sending it to one side or the other; at *B* is a water pipe with a tap for each trough; at *C* there are riffles about 4 inches deep, and at *D* there are slides about 6 inches broad and the full width of the trough, which can be lifted by the handles *f*. The dirt which accumulates behind the

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[1] *Trans. Min. Inst. Scotland*, Vol. XI. p. 216.

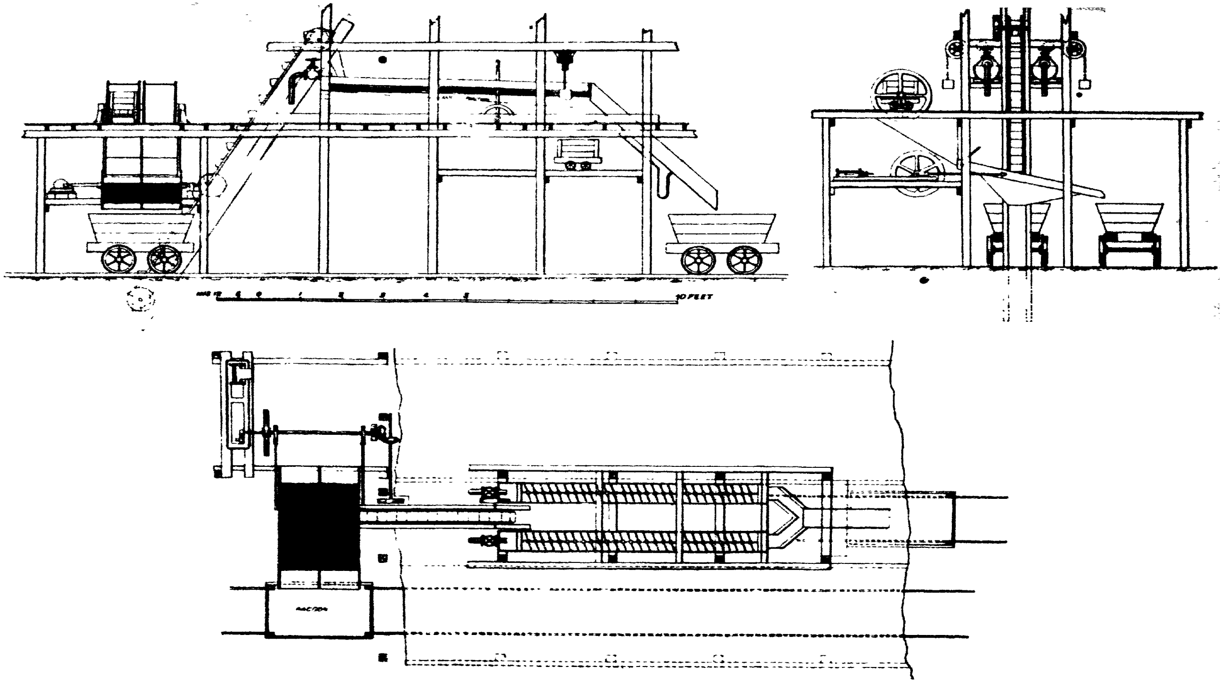

Fig. 239. Bell and Ramsay washer. Plan, side and end elevation.

riffles *C* is removed by lifting out the slides and allowing the dirt to drop into a lower trough, down which it is carried by a stream of water to *E*, where it is loaded into tubs; the washed coal is discharged at *G*. Whilst one trough is being cleaned out, the other one is in use. This appliance washes about 10 tons of coal per hour, removing about 20 per cent. of dirt; the cost of labour is given at 2·25*d*. per ton of coal treated.

In the **Bell and Ramsay**[1] **Washer**, Fig. 239, the trough is made of iron, is semicircular in cross-section, and is hinged at the upper end; along it run stirrers attached to a shaft which gives them a vibrating motion to and fro. The trough is fitted with riffles, some of which are fixed, whilst others can be moved by means of levers, as shewn in detail in Fig. 240. When the trough has to be cleaned out from the accumulated dirt, the bottom end is lowered clear of the stirrers and fixed riffles, whilst the movable riffles are lifted by means of their levers. The dirt is then flushed out by a stream of water into a waggon placed to receive it, and the trough is then lifted back into its place again and the washing operation continued.

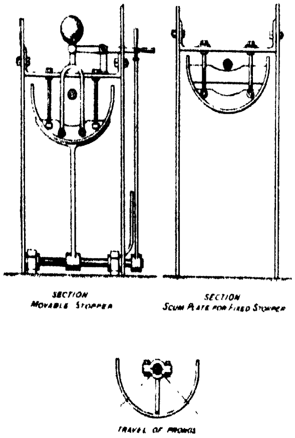

Fig. 240.
Riffles of Bell and Ramsay washer.

Various improvements have been made in trough washers with the object of rendering the action continuous and thus doing away with the intermittent method of working two troughs alternately. This can evidently be accomplished by some method of removing the dirt continuously at the upper end, whilst the clean coal flows off at the lower end. Of the various methods in use we may consider travelling scrapers, a travelling trough and a trough with screw conveyor.

The **Elliott Washer** (Figs. 241 and 241^a) consists of an iron or steel trough, about 18 inches wide at the top and 12 inches at the bottom, as shewn in Fig. 242, about 12 inches deep and 60 feet long, set at a gradient of from 1 in 12 to 1 in 18. Above each end of the trough there are sprocket-

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[1] *Trans. Min. Inst. Scotland*, Vol. XI. p. 185.

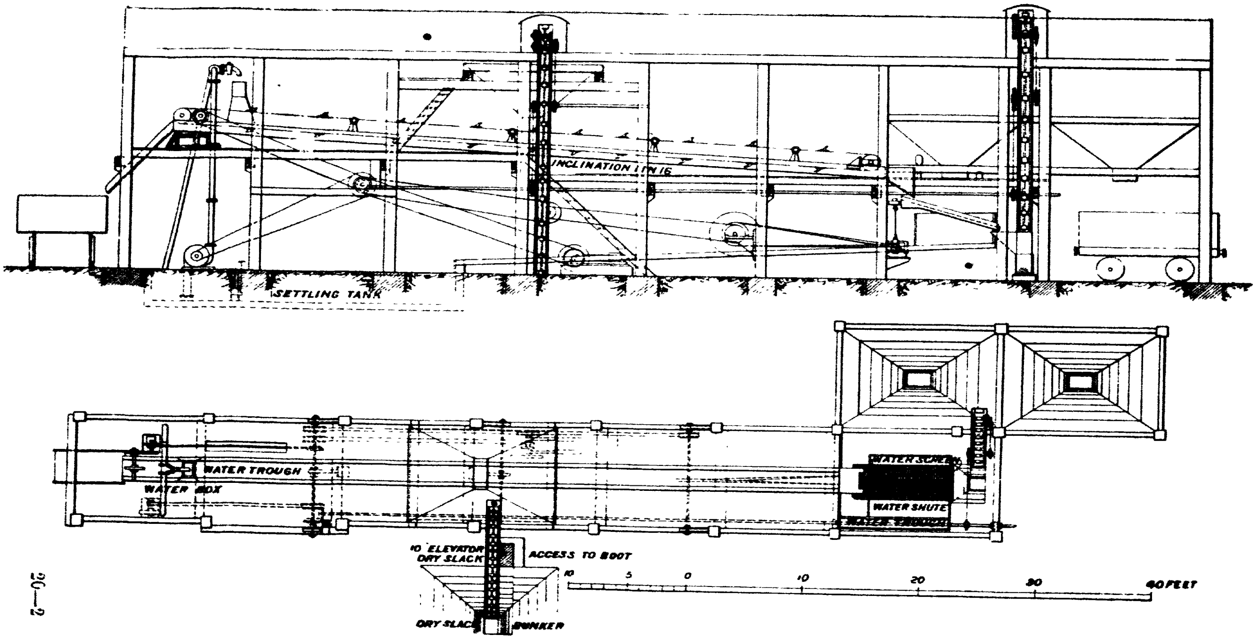

Fig. 241. Elliott washer. Plan, and side elevation.

wheels driving endless chains, to which scrapers, about 3 inches deep, are attached; these scrapers act as riffles, but as they slowly travel up the trough, they carry the dirt up with them and discharge it at the upper end of the trough, whilst the washed coal flows off at the lower end. In order to clean the dirt from all particles of coal, the coal to be washed is fed in about 20 feet from the top end of the trough, so that the dirt travels for this distance in clear water only and is thus thoroughly washed. Each trough treats from 50 to 100 tons of coal per day; a larger output is dealt with by increasing the number of troughs. The figure (Fig. 241) shews a single trough plant capable of dealing with 100 tons per day. At the New Consolidated Charlotte Pit, in Silesia[1], an Elliott washer, 60 feet long, with a gradient of 1 in 15, washes

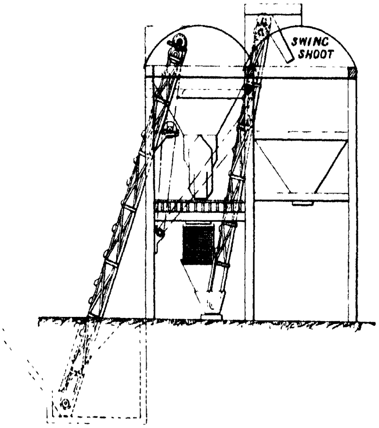

Fig. 241*. Elliott washer. End elevation

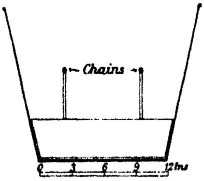

Fig. 242. Trough of Elliott washer. Cross-section.

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[1] *Zeitsch. f. Berg. Hütt. u. Salin. Wesen*, Vol. XLV. 1897, B. p. 233.

60 tons of coal per day, which has passed through a screen of 0·275 inch mesh, with a water consumption of 175 gallons per minute. The dirty coal contains 15 per cent. of ash, and the washed coal only 5 to 6 per cent., whilst the dirt removed contains 50 to 60 per cent.

At Wirral Colliery, Cheshire[1], an Elliott washer has been erected, the trough being 60 feet long, 1 foot 6 inches wide at the bottom, 2 feet 6 inches at the top, set at a gradient of 1 in 12; the scrapers are 4 inches deep and set 6 feet apart; the coal is fed in 30 feet from either end of the trough. This washer can deal per hour with 10 to 11 tons of coal that has passed a 1-inch screen. The unwashed slack contains 25 per cent. of ash, whilst the washed coal and the dirt contain respectively 4·20 per cent. and 68·75 per cent. of ash. The complete plant costs £730, and an additional trough has been put in, thus doubling the capacity of the plant, for another £260. The cost of washing is given as about 0·875*d.* per ton.

The Elliott washer is in use at several collieries in Great Britain, North of France, etc.

A very similar trough washer, built by Messrs Inglis and Hossack of Airdrie, is in use at the Longrigg Colliery, Slamannan[2]; it is shewn in Fig. 243, and consists of a trough 60 feet long, 2 feet wide, and 11 inches deep, set at a gradient of 1 in 18; the coal to be washed, together with some water, is fed in about 15 feet from the top end, and the remainder of the water is run in at the very top of the trough. In the bottom of the trough lies an endless chain to which are attached scrapers 2 inches deep, about 4 feet apart, the chain returning as shewn beneath the trough. This machine washes efficiently some 200 tons of coal per day.

It will be seen that this appliance is practically identical with the last-named, differing only in the position of the endless chain.

The **Murton Washer** (Wood and Burnett's patent)[3] consists of a trough composed of segments like a picking belt, but fitting watertight against each other, as shewn in Fig. 244. The trough thus formed is 60 feet long, 3 feet wide and 8 inches deep, and it is set at a gradient of 1 in 18. The joints of the plates, which are 3 feet long, are so arranged as to form riffles about 2 inches deep. Coal, with a certain amount of water, is fed in at 12 feet from the upper end, the remainder of the water being supplied close to the top end. The trough travels

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[1] *Trans. Fed. Inst. Min. Eng.* Vol. XI. p. 55.

[2] *Trans Min. Inst. Scotland,* Vol. XI. p. 218

[3] *Trans. Fed. Inst. Min. Eng.* Vol. IX. p. 42.

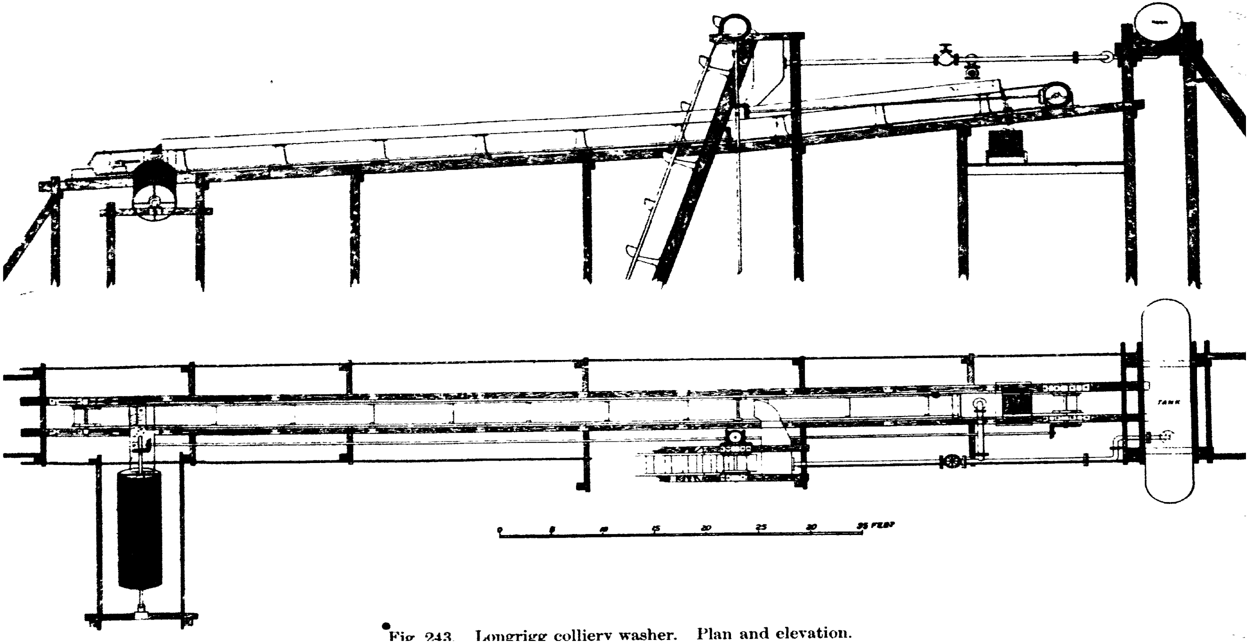

Fig. 243. Longrigg colliery washer. Plan and elevation.

upwards at a rate of 8 to 10 feet per minute, carrying the dirt with it, whilst the water carries the clean coal downwards and discharges it at the lower end. At Murton Colliery such a washer treats per day 400 tons of coal that has passed through a ⅜-inch screen, and uses 450 gallons of water per minute. The power required to drive the washer proper is about 1½ H.P. The dirt washed out amounts to 16 or 17 per cent., and the washed coal contains 2·86 per cent. of ash. The cost of washing is given as 0·76*d*. per ton of slack, and the first cost of the plant as £2000.

A similar machine at Nunnery Colliery deals with 22 tons of slack per hour.

The **Blackett Washer** (Blackett and Palmer's patent) is shewn in Fig. 245. The trough is here replaced by an iron cylinder, to the inner surface of which is riveted a spiral strip of iron; this latter acts simultaneously as a series of riffles, and as a screw conveyor, carrying the dirt forward and discharging it at the upper end of the cylindrical trough.

Blanket Strakes are scarcely used, except for concentrating the auriferous pyrites and other heavy minerals contained in crushed gold quartz. They consist of troughs 12 to 20 feet long, 2 to 3 feet wide, and about 6 inches deep, set at gradients that may range from 1 in 6 to 1 in 16. In the bottom of the troughs are laid blankets from 4 to 6 feet long, of the same width as the trough; they must be laid like tiles on a roof, the lower edge of each overlapping the upper edge of the next lower one for a distance of some

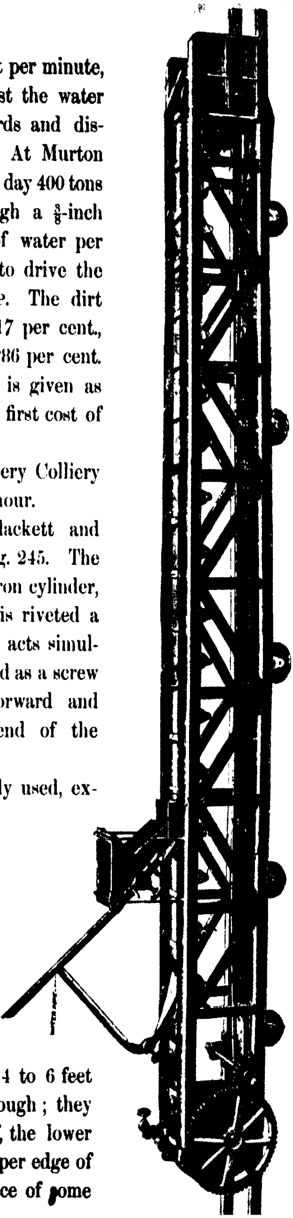

Fig. 244. Murton washer. Perspective.

6 inches. The pulp to be concentrated is run in, best over a headboard fitted with pins, so as to produce a stream of uniform depth. The heavier mineral sinks to the bottom and, becoming entangled in the nap of the blankets, is retained thereby, whilst the lighter minerals flow off in the stream of water. After a certain time the nap of the blankets becomes filled with the accumulated concentrates; the stream of pulp is then diverted into an adjoining strake, and the blankets are folded up and carried to a wooden tank filled with water, in which they are washed, until freed from the accumulated mineral. The latter is thus collected in the tank, from which it is removed by hand for further treatment. Blanket strakes are worked usually in pairs, but sometimes in threes, two filling up, whilst the third is being washed. Instead of blankets, sacking, bass matting, plush or similar materials may be used. Blanket strakes were very extensively employed at one time, but are now but little used, having been generally replaced by mechanical concentrating appliances. Their working cost is high, both for labour and for material, but they are very easily and cheaply constructed, and are therefore still used at times in small and preliminary installations.

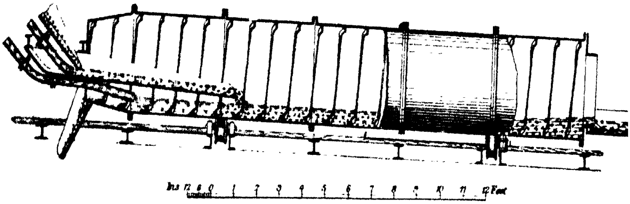

Fig. 245. Blackett washer. Sectional elevation.

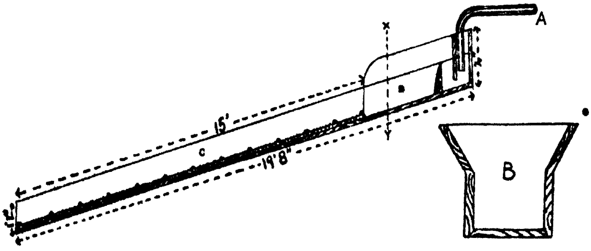

Fig. 246. Peat strakes for platinum. Longitudinal section and cross-section on XY.

A very similar appliance, shewn in Fig. 246, is used for concentrating platinum sands in the Urals[1]. It consists of a box B of the section shewn, about 1 foot wide, lined with sheet iron; at the upper end a stream of water is introduced through a pipe *A*, and is kept at a constant head by means of the baffle boards as shewn. This box opens into a trough *C*, also about 1 foot wide and 15 feet long, set at a steep gradient; this is floored with well-washed peats, kept down by cross-pieces of wood, held in their place by wedges and forming shallow riffles. The sands to be treated are thrown into the box *B* in small lots and worked about with a hoe or narrow shovel; most of the coarser platinum remains in *B*, and the rest is caught by the riffles and peats in *C*. The peats are taken out from time to time, washed and beaten, and the material retained by them is thus collected.

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[1] *The Mineral Industry*, Vol. VI. p. 549.

Canvas tables may appropriately be considered here, although they are very often used for fine slimes and worked with thin films of water, and should therefore in such cases be included with the next series of appliances. They consist of tables or shallow troughs, over which a piece of canvas is stretched; very often the canvas is nailed to a light wooden frame which is laid on the table. They are set at gradients ranging from ⅛ inch to 1½ inches per foot according to circumstances. In some cases narrow troughs, 18 inches wide or a little more, are used, in other cases tables from 4 to 12 feet wide. The length may vary from 10 feet to 100 feet. The conditions under which they are worked thus vary very widely, but they are almost exclusively used for concentrating auriferous pyrites out of the pulp produced by stamping gold quartz. In Australia three tables, 4 to 5 feet wide and 80 to 100 feet long set side by side, on two of which the concentrates are being deposited whilst the third is being cleaned, are considered sufficient to treat 100 tons of crushed quartz per 24 hours, each table taking about 30 cubic feet of pulp per minute. In America wide tables are used at times, but more often narrow strakes; a strake, 20 inches wide and 40 feet long, will treat from 1 to 3 tons of crushed material in 24 hours. The canvas in these tables wears well, and the chief expense is for labour. The rough surface of the canvas checks the flow of the lowermost film of water considerably, and thus helps to catch very fine slimes. These tables can therefore be used to treat pulp carrying only a very small percentage of finely divided heavy material, and they are often arranged to follow other appliances so as to collect the last portions of valuable material in the pulp. The concentrates are either brushed off them or washed off with a hose, and are rarely clean enough to dispense with further concentration.

The principle of separation in shallow streams of water in its pure form is especially applicable to slimes and very fine sands, the increments in the velocity of successive water films being greater when thin films are considered than when these are deeper. Hence appliances working on this principle can treat particles far too fine to be separated by rate of falling in water. In their simplest form the appliances to be considered consist of inclined planes upon which a given quantity of the mineral pulp to be treated is fed, the quantity of water and the inclination of the plane being such that the heavier particles either remain at rest or are carried very slowly down the plane, whilst the lighter particles are carried down more rapidly. After a given interval the supply of pulp is cut off and a stream of clear water run on, still in such quantity as not to wash off the heavier mineral, but so as to thoroughly clean it by removing all the lighter particles; sometimes this action is assisted by gently rubbing or brushing the heavier residue upwards against the current so as to get rid of lighter particles mechanically entangled in the heavier; finally the clear concentrated heavier particles are washed off or brushed off into a suitable receptacle, and the entire process recommences. The operation is thus intermittent, with three separate stages of depositing, cleaning and washing off.

Professor Richards[1] has investigated the best conditions of angle of slope and quantity of water to effect a complete separation between grains of mineral of given specific gravities for various sizes of the same; his results appear to be embodied in the following table:

For coarse	pulp	the best	slope is	2° 45′	with	12 lbs.	of water	per foot	of width
For medium	,,	,,	,,	5 6	,,	5 6 lbs.	,,	,,	,,
For finest	,,	,,	,,	8 10	,,	2 lbs.	,,	,,	,,

There is ample room here for further experimental investigation, the nature of the surface of the plane being one of the most important conditions affecting the efficiency of the appliance; it seems to be pretty clear that for good work the particles should roll and not slide down the plane, a condition determined essentially by the shape of the particles and the nature of the surface.

The typical appliance for this operation is known as the **Frame**; it is sometimes spoken of also as the Flat Buddle, but this name is better restricted to the appliances already described on p. 293. It consists usually of a table of well-planed boards 1½ inch thick, secured to a substantial wooden frame; it is 12 to 20 feet long, 3 feet 6 inches to 5 feet wide, set at a gradient ranging from 5° for slimes to 12° for fine sands. It is fitted with a headboard with regulating pins, the inclination of the headboard

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[1] *Ore Dressing*, Vol. II. p. 707 and *Trans. Amer. Inst. Min. Eng.* Vol. XXVII. p. 76.

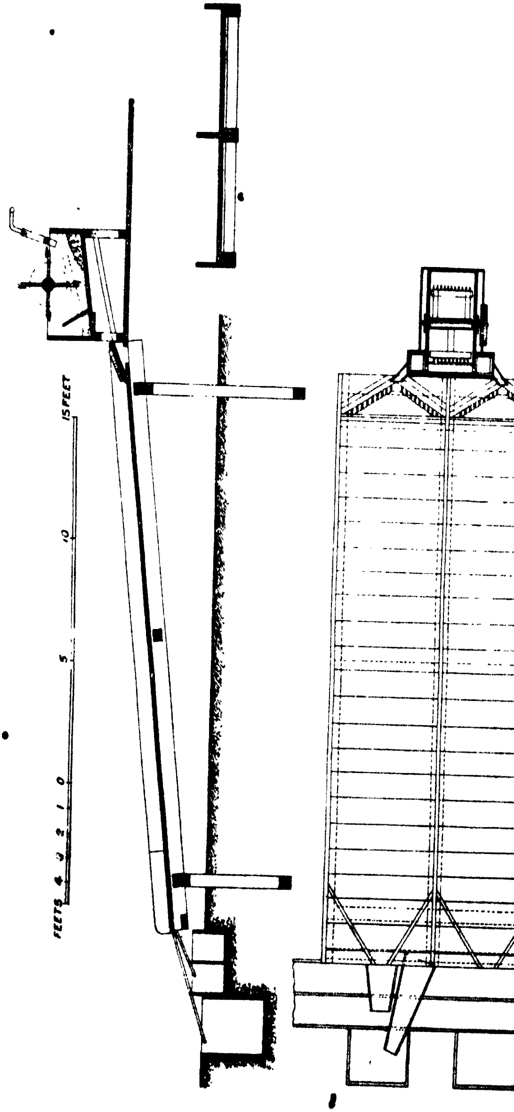

Fig. 247. Frame. Plan, longitudinal and cross-section.

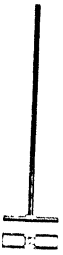

Fig. 248. Hoe for frame.

being about twice that of the frame itself. These frames are always worked in pairs and are in fact usually built double. The general construction of such a double frame is shewn in Fig. 247, after Kirschner[1].

According to Rittinger a frame 4 feet wide and 12 feet long will treat 0·08 to 0·12 cubic foot of pulp carrying slimes, or 0·3 to 0·5 cubic foot of pulp carrying fine sand per minute, the pulp in the former case containing 5 to 8 lbs. of dry material and in the latter 15 to 18 lbs. per cubic foot of pulp. The total capacity is 2 cwt. of the former class or 15 cwt. of the latter class, or say on the average 6 cwt. of mineral per 10 hour shift; the quantities of water in the pulp for depositing, of water used for cleaning the concentrates, and of water for washing it off, are in the proportions of 1 : 1 : 3, and the periods of time occupied in the respective operations are about 4 minutes, 2 minutes, and 2 minutes. In working the frame, the pulp is allowed to run over one of the divisions of the table in a thin film, when the heavier portions remain upon the table, whilst the lighter portions flow off with the pulp. In order to keep the surface smooth and to prevent channels forming, the deposit is lightly rubbed from below upwards with the wooden hoe shewn in Fig. 248. When a sufficient layer of concentrates has been deposited, the pulp is diverted to the neighbouring table, and a thin stream of clear water run on; the tailings washed off by it usually contain some of the heavier portions and are collected separately, forming middlings to be washed over again. The heads that then remain on the table are practically clean and are washed off into a special receptacle, a broom being often employed to sweep them down. From this is derived the German name of the appliance (*kehrherd* = sweeping table). The old Cornish form of this appliance is a table resting on pivots, so that after the mineral has been deposited, the table can be tipped up into a vertical position, and the deposit washed off by means of a perforated pipe running above it. This arrangement is shewn in Figs. 249, 250 and 251[2], in which the method of working the frame and that of tipping it up to wash off the heads are clearly shewn, the latter operation being performed by turning over a triangular trough of water suspended above the frame.

The chief modifications of this simple appliance are variations in its dimensions and proportions. The frame used in the Welsh[3] Lead Mines

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[1] *Grundriss der Erzaufbereitung*, L. Kirschner, p. 84.

[2] *Proc. Inst. C. E.*, "On Dressing Tin and Copper Ores in Cornwall," by Henderson, Vol. XVII.

[3] *Mem. Geol. Survey*, "On the Mining District of Cardiganshire and Montgomeryshire," by Warington W. Smyth, Vol. II. Pt. II. p. 675.

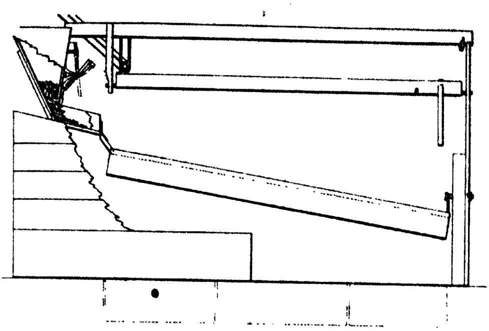


Fig. 249. Frame in position for receiving mineral. Side view

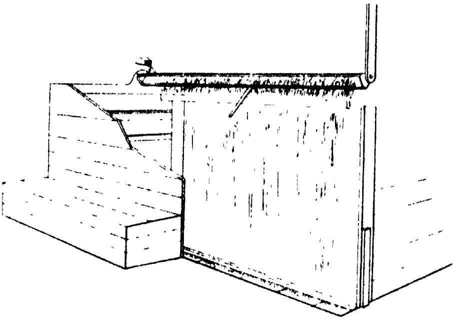
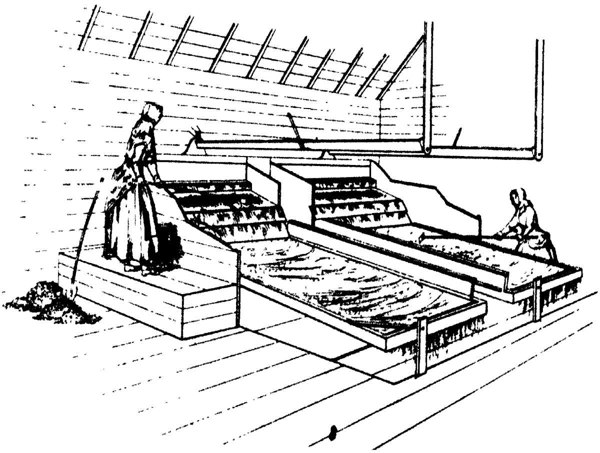


Fig 250. Frame tipped up for washing off deposited mineral. Perspective



is very broad in proportion to its length, namely 10 feet wide by 7 feet long, with a fall of 2½ inches in that length. A gentle stream of water flows over the surface and the ore to be dressed is placed in a small heap on one side of the water supply and is drawn with a hoe partly against and partly across the stream to the other side of the table.

Whilst some frames, as in Figs. 249 to 251, are carried on trunnions placed in the centre line of their length, so that when the mineral has been deposited and cleaned, the frame can be turned vertical, others are turned upside down for the more ready removal of the clean concentrates. Occasionally such a frame consists of two tables back to back, so that the mineral may be washed off the table that is undermost whilst it is depositing upon the other table. At times three tables have been arranged with their longer sides fastened together so as to form a triangle in cross-section, the whole structure being attached to a central shaft, as indicated on the subjoined diagrammatic cross-section, Fig. 252, where *a*, *b*, *c* are the three tables and *d* the central supporting shaft. The mineral is deposited and cleaned on the table, such as *a*, which happens to be uppermost at the time, and when ready the structure is turned through 120°, thus bringing another table, such as *b*, into position for depositing mineral upon it, whilst the deposit on *a* is washed off into a trough beneath.

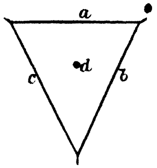


Fig. 252. Triangular frame. Diagrammatic cross-section.

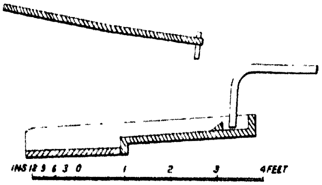


Fig. 253. Small frame for platinum. Vertical section.

A very small frame is used in the Urals for finishing the washing of partially concentrated platinum sands[1]; it consists of what are really two tables separated by a drop of 2 inches, the two together being 3 feet wide and 4 feet 9 inches long. A stream of water runs on to the upper table upon which the sands to be treated are thrown; they are worked about on this surface by means of a little wooden hoe 8 inches wide, until they are fairly clean, when they are allowed to be carried on to the lower table by the stream of water, and the washing is there finished, the clean platinum sands being finally collected off both tables. It will be noticed that in this modification the stages of depositing and cleaning succeed each other without any break at all, and form in fact only one operation.

The constant attention needed for these appliances together with their relatively small output renders their operation costly, and many devices have been resorted to in order to render them automatic. A very ingenious one is the **Self-acting Frame**[2], used in Cornwall for fine tin ores, shewn in Fig. 254. It consists of two tables one about 5 inches above the other, both set at a slope of about 1 in 7. The pulp is delivered through the launder *A* on to the headboard *B* (Fig. 254) and thence on to the upper table, over which it flows, traversing next the lower table, the clean ore being deposited upon these two tables, whilst the waste flows off at the bottom of the lower table into the launder *C*. There is no provision for cleaning the deposit, but the washing off device consists of two triangular troughs *D*, *D*, pivoted along the apex of the triangle, which gradually fill with clear water from the launder *E*. These triangular troughs are so arranged that when full they overbalance, and the water pouring out in a heavy stream flushes the tables. By the system of levers shewn, the act of tipping up these troughs lifts the covers from two troughs *F*, *F*, one below each table, into which the concentrated tin ore is washed. The larger quantity and the cleaner part of the tin ore is deposited on the upper table, hence the contents of the two are kept separate. As soon as the troughs *D*, *D*, have discharged their contents they are brought back to their original position by the counterpoise *G*, when the operation of depositing recommences. The upper section in Fig. 254 shews the frame during the depositing stage, and the lower section during the flushing stage. By this arrangement one lad can attend to 20 frames, whilst the hand-worked frames need from one to two lads to each frame.

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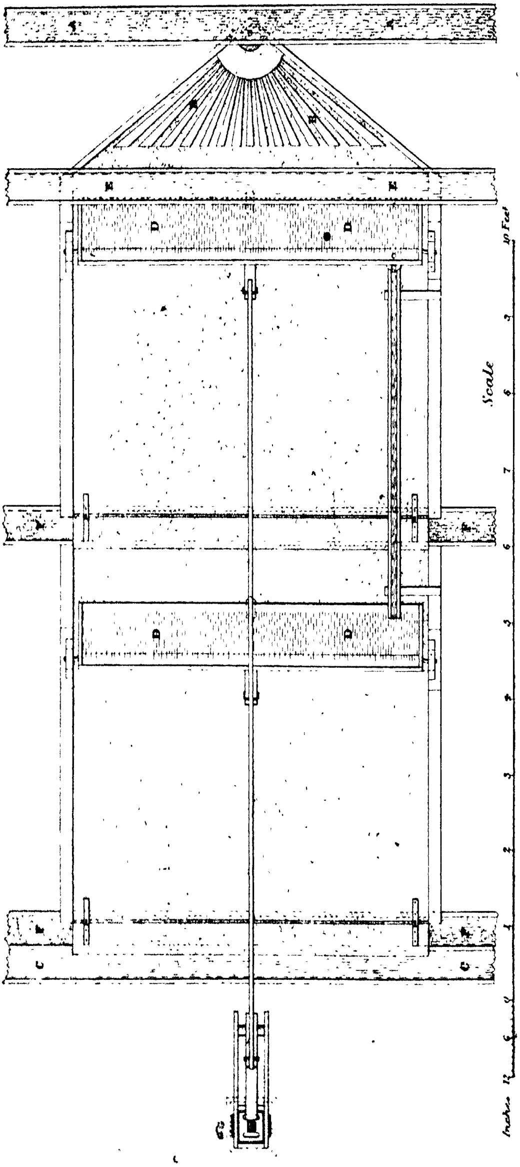
[1] *The Mineral Industry*, Vol. VI. p. 550.

[2] *Proc. Inst. Mech. Eng.*, "On the Mechanical Appliances used for Dressing Tin and Copper Ores in Cornwall," by Ferguson, 1873, p. 119.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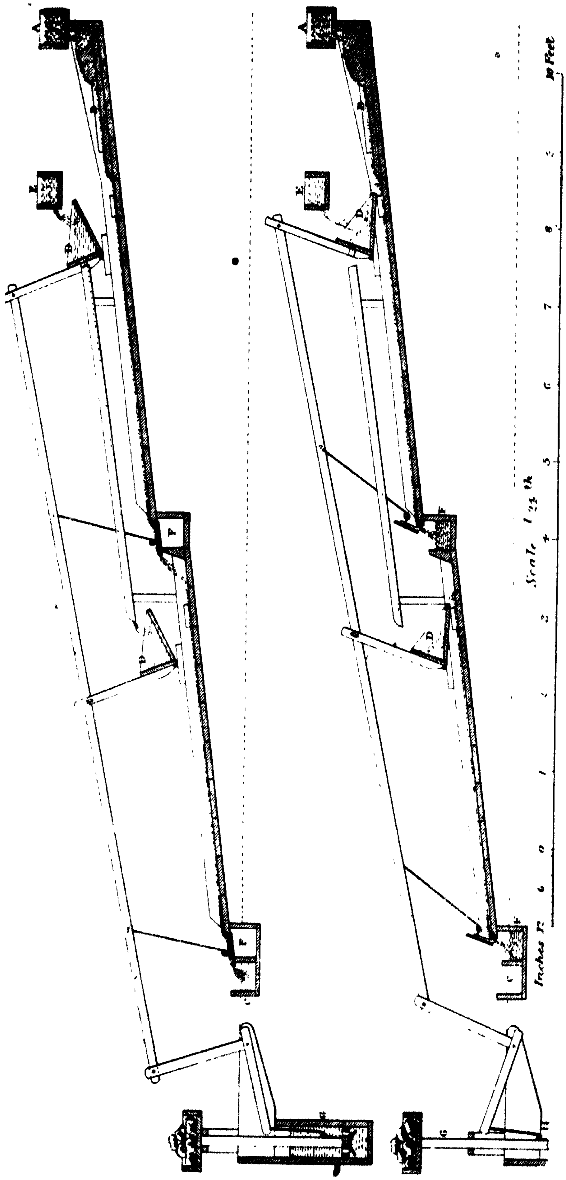

Fig. 254. Self-acting Cornish frame. Plan and sectional elevation.

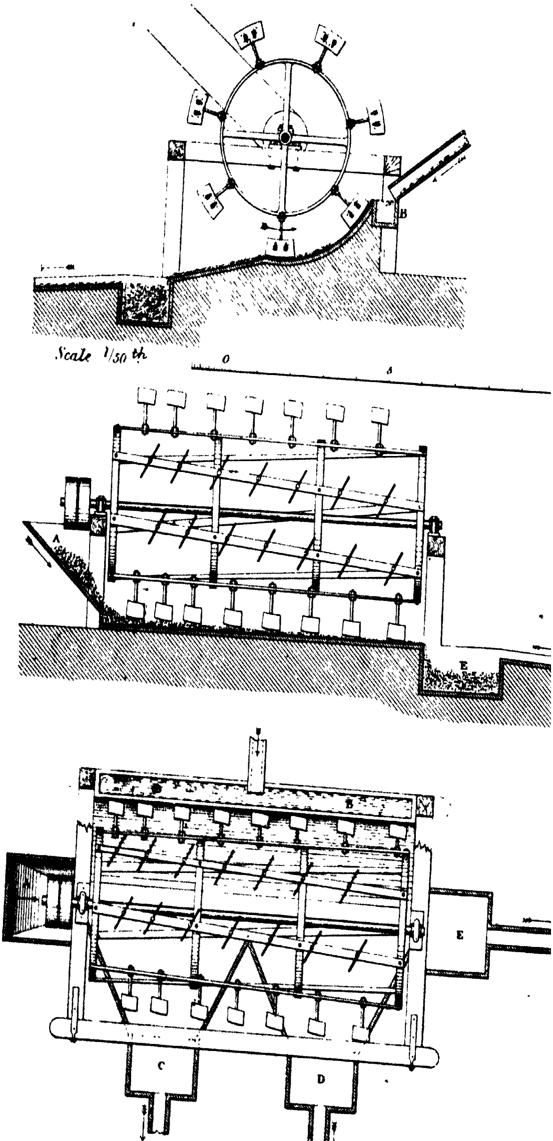

Fig. 255. Propeller knife buddle. Plan, vertical and cross section.

Another automatic appliance, used chiefly for fine lead ores, but which has also been used for fine tin ores, is the **Propeller Knife Buddle**[1], shewn in Fig. 255. The frame proper consists of a table, made either of plank or cement, or lined with sheet iron, finished accurately to the shape of about one quarter of a hollow cylinder, as shewn in the section. Above this a light iron frame about 6 feet in diameter rotates on a horizontal axis, carrying a number of scrapers or knives arranged in continuous spiral lines round the frame, so as very nearly but not quite to touch the concave table, the frame making about 20 revolutions per minute. The pulp to be dressed runs in at one end from the hopper *A*, whilst a stream of clear water is supplied from the trough *B* along the whole of the upper edge of the concave table; the heavier particles deposit on the bottom of the table, the lighter waste being washed by the water into the hutches *C* and *D*; the action of the revolving scrapers carries the heavier portion along gradually, being exposed all the time to the cleaning action of the water, until it is discharged well cleaned into the hutch *E*. The contents of the first waste hutch *C* require dressing again, those of the hutch *D* can be allowed to run to waste.

The knife buddle is still used a good deal in the North of England for cleaning fine lead ore obtained from the ordinary buddles, etc.

An appliance important even more on account of its subsequent developments than on its own account is **Brunton's Cloth**[2], introduced about the year 1840, and still used for fine lead ores. It is shewn in Fig. 256, and consists of a wooden frame with a roller at either end, over which runs an endless belt of canvas stiffened with several coats of paint; laths or small rollers are fixed at intervals across the frame so as to keep a flat upper surface. The upper roller is driven by gearing and causes the belt to move upwards at the rate of about 15 feet per minute. The pulp to be treated is delivered on to the travelling belt at about one-third of its length from the top, whilst a supply of clear water runs on to it close to the top roller.

Owing to this arrangement the films of water in immediate contact with the belt are not only prevented by friction from flowing downwards, but are actually carried upwards by the motion of the belt, whilst the upper films of water flow downwards at a rate regulated by the angle of inclination of the belt and the total quantity of water supplied to it.

The lighter portions of mineral run down the belt and flow off at the lower edge; the heavier portions sink down on to the belt and, being carried upwards by it, pass through the stream of clear water, which removes any of the adherent lighter portions, and are finally carried over the top roller and washed off in a trough suitably placed into which the cloth dips. These machines are practically automatic, one lad being able to attend to several; they are cheap to construct, take very little power to work them, and are fairly efficient; on the other hand they require a good deal of water, and their capacity is somewhat limited.

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[1] *Proc. Inst. Mech. Eng.*, "On the Mechanical Appliances used for Dressing Tin and Copper Ores in Cornwall," by Ferguson, 1873, p. 119.

[2] *Proc. Inst. C. E.*, "The Dressing of Lead Ores," by T. Sopwith, Vol. xxx. 1870.

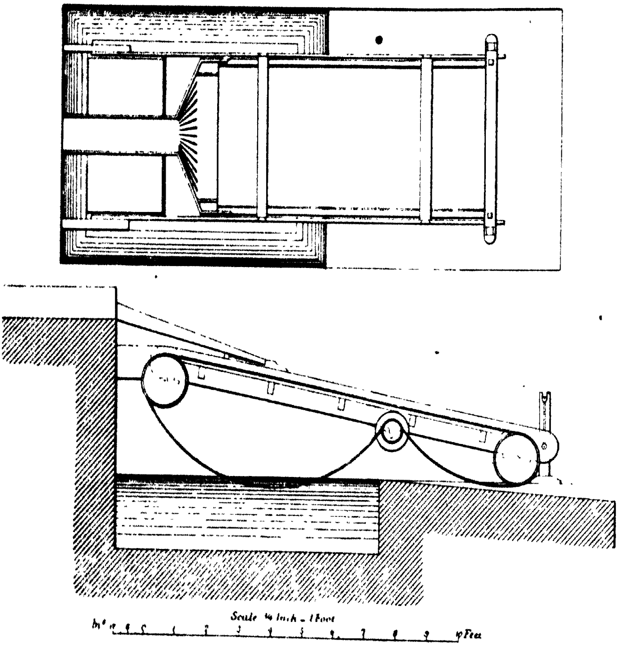

Fig. 236. Brunton's cloth. Plan and elevation.

A number of important machines known as Vanners, are merely an extension and development of the principle of the Brunton cloth; these will be fully discussed in the next chapter. Those interested in the history of these inventions will find a good account of it in a paper by F. Schmidt[1] upon endless belt appliances for concentrating minerals.

The **Ferraris Belt** differs from the foregoing in that the belt has a motion at right angles to the stream of water instead of parallel to it. A perspective view of it is shewn in Fig. 257[1]; it consists of a rubber belt 27 inches wide carried over two drums, the centres of which are 13 feet apart, and supported upon intermediate rollers at every 2 feet. One of the drums is driven by friction gearing so as to give the belt a speed of about 20 feet per minute; the shaft of the other drum rests in sliding bearings and keeps the belt in proper tension. Both drums and rollers have a uniform slope from one side (the back side) to the front, so that the belt takes this same slope. The pulp is fed on to the belt at its upper edge, close to one end (i.e. the following end); as it moves along it is subjected to the action of a number of jets of clear water. The lighter portions of the pulp flow off at once, the heavier remaining on it and being carried along by the motion of the belt. They are thus submitted to the action of successive jets of water, which wash them off into various tanks, the heaviest portions resisting this washing action for the longest time and being therefore the last to leave the belt. Such a belt can thus produce a number of different grades of concentrates as well as barren tailings, whilst the Brunton

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[1] *Bull. Soc. Ind. Min.* Vol. VIII. Ser. 3, 1894, p. 641.

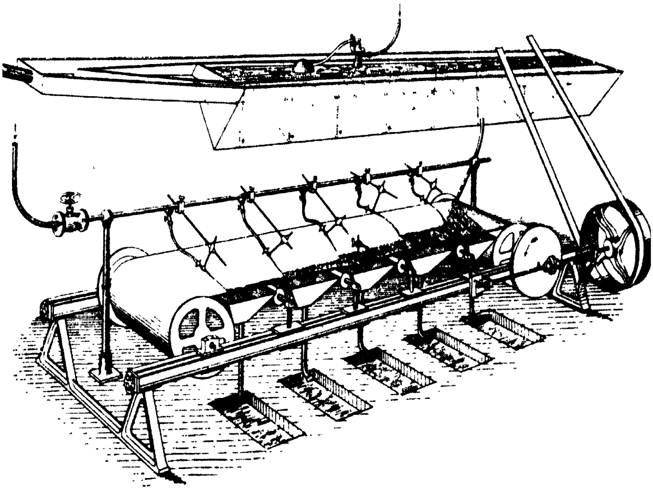

Fig. 257. Ferraris belt. Perspective.

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[1] *Oesterreichische Zeitsch. f. Berg. u. Hütt.-Wesen*, "Dressing of Slimes," by E. Ferraris, Vol. XLII. 1894, p. 421.

Belt makes concentrates and tailing only. At Monteponi such belts dress lead ores in the state of fine sands of less than 0·06 inch mesh. Each belt will take about 1·5 cubic feet of pulp and about 2 cubic feet of clear water per minute; it will treat about 3 tons of sands in an 11 hour shift, and requires only one lad to attend to it. It is easily and cheaply built, and its working is said to be perfectly satisfactory.

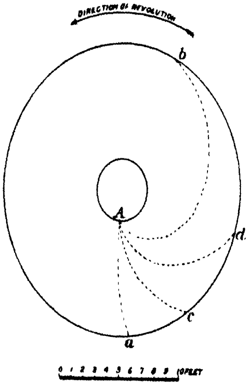

Fig. 258. Diagram of action of revolving slime table.

The **Revolving Slime Table** is known by various names, such as the slime buddle, the Zenner buddle (in the North of England after its introducer), and occasionally as the Harz table; the term buddle, as already pointed out, is better not applied to machines of this type, but restricted to those working coarser sands in a deep water current. This table bears the same relation to the flat table or frame as the round buddle does to the box buddle, and may be looked upon theoretically as consisting of a number of flat frames set radially. It consists of a round table, the surface of which has the form of a very flat cone, which revolves slowly on its (vertical) axis; a stream of pulp runs down a narrow segment, whilst streams of water of different strengths are allowed to run over the rest of the surface. If the pulp contains particles of different specific gravities their rates of travel down the table will be different. Whilst their motion as referred to the table will always be in radial straight lines, their actual motion in space will be compounded of this motion and of the slow motion of rotation of the table, and their paths will be spirals; particles moving rapidly will follow a path such as *Aa*, those moving very slowly a path such as *Ab*, whilst those having intermediate rates of travel down the table will follow paths such as *Ac*, *Ad* (Fig. 258). Slimes can thus be efficiently separated in accordance with the specific gravities of their component particles, and the rotating table is accordingly a very generally used appliance for dressing slimes. In practice these tables range from 10 feet to 25 feet in diameter, the inclination of the conical surface being from 5° to 8°, the larger tables and flatter slopes being employed for the more finely divided slimes; they make a complete rotation in from 1 to 4 minutes and require from ½ to 1½ H.P. The pulp to be concentrated carries from 4 to 7 lbs. of dry solid matter per cubic foot, and a table will treat from 5 to 20 tons in 24 hours; on an average a table takes 1½ to 3 cubic feet of pulp per minute, and about an equal amount of clear water for cleaning and washing off, the former of these operations taking about ⅔ and the latter about ⅓ of the total quantity of clear water. It will be understood that there is necessarily a great range in these general figures, because the capacity and mode of operation vary greatly according as fine slimes or coarser sands are being treated and as the minerals to be separated differ greatly or only comparatively little in specific gravity.

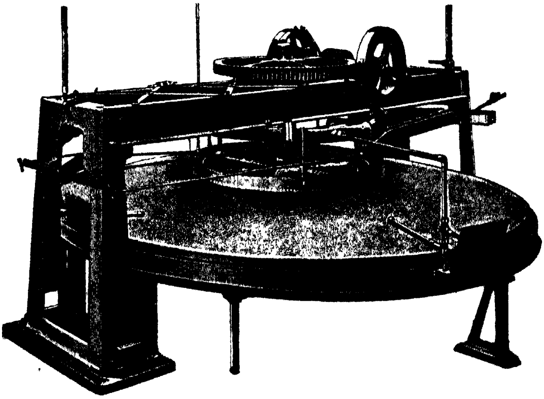

Fig. 259. Revolving slime table. Perspective.

These tables are usually built upon an iron framework, the surface consisting usually of segments of wood well planed and finished, or else of cast iron planed or of sheet iron covered with a thin layer of cement. Such a table as built by the Humboldt Engineering Works is shewn in Fig. 259. The table is driven by a tangent screw and worm wheel keyed to the upper end of a vertical shaft. This latter carries a light iron framework, supporting the table which is either made of corrugated iron covered with a layer of cement, or else of planed cast iron.

There is a circular funnel-shaped distributor, into which the pulp is conveyed by a trough, and which allows the pulp to run in a thin stream over a segment of the table; the aperture of this segment can be varied by adjusting the distributor, and is usually from 90° to 180°, averaging usually about 135°; the tailings run straight down over the table and are caught in a corresponding segment of the channel shewn surrounding the table. The material next passes under a series of jets which are attached to a branch pipe coming from the annular pipe shewn as surrounding the shaft; the washing off jets are attached to a separate pipe branching off the same annular pipe. The circular channel surrounding the table is divided into three (more rarely four) divisions by adjustable partitions, making as many different products, namely, concentrates or heads, middlings (one or two classes), and practically barren tailings. The following figures give the leading particulars for a medium size table:

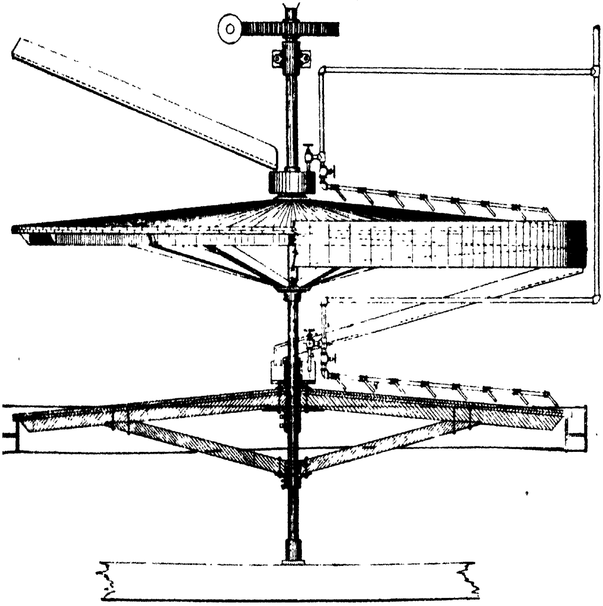

Fig. 260 Double slime table. Sectional elevation.

Diameter of table	16′ 0″
Revolutions of table per hour	80
Power required	½ H.P.
Consumption of clear water per minute	40 gallons
Capacity, according to nature and fineness of slimes, per 24 hours	4—8 tons.

Sometimes these tables are fitted with brushes to brush off the concentrates instead of washing them off, so as to economise water or to avoid excessive dilution. When economy of space is particularly important for any reason, multiple tables, in two or three decks, such as shewn in Fig. 260, may be constructed; they are, however, cumbersome and difficult of adjustment and their use is not to be recommended if it can be avoided. They are usually built of wood on light iron frames: they are best made, as shewn in Fig. 260, of narrow boards in two layers, breaking joint.

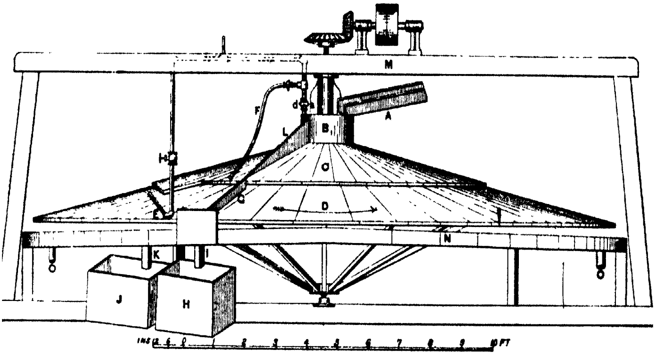

Fig. 261. Evans slime table. Elevation.

The **Evans Slime Table**, as built by Messrs Fraser and Chalmers, Ltd., is shewn in Fig. 261. It consists of a wooden table built of segments of pine wood upon an iron "umbrella" frame. The table is furnished with a wide headboard *C*, cut into a spiral form so that the concentrates are protected by the widening portion from the action of the wash-water during the greater part of the travel of the table. The pulp is delivered through the trough *A* and distributor *B* on one side (the right-hand side in the figure) of the vertical division *L* that rests upon the headboard; it runs on to the revolving table *D*, and the tailings run off directly into the circular launder *N*, and are discharged through the pipes *O*. The middlings are cleaned and washed off by the water jets from the perforated pipe *E* into another division of the launder, and are discharged through the pipe *K* into the box *J*. Finally the concentrates which have been hitherto shielded by the projecting portion of *C* are washed off by the pipe *F* into a short section of the circular launder, and thence through the pipe *I* into the box *H*, being directed into this course by the dividing board *G*. The headboard *C* is suspended from the crossbeam *M* so that its position can be adjusted relatively to that of the table as may be required. This table makes 1 revolution in 80 seconds. It is 19 feet in diameter; the headboard on the feed side is about 7 feet in diameter, and this diameter increases gradually on the clear water side to about 10 feet in diameter; the table slopes 1¼ inches and the headboard 1⅝ inches to the foot. Its capacity is 25 to 30 tons per 24 hours.

A **Concave Table**[1] on the same principle as the revolving slime table is sometimes used, bearing the same relation to the last-named that the concave buddle does to the convex buddle. One form of it, as used at the Moonta Mines, S. Australia, is shewn in Fig. 262.

Shranz[2] has introduced a table with two headboards diametrically opposite each other on large tables 16 feet to 18 feet in diameter; he uses curved perforated clear water pipes for spraying the products off the table. By these means a high capacity is attained, but the consumption of water is considerable. At Mühlenbach such tables with surfaces of cast iron and of cement are in use for treating lead and zinc ores; they make 1 revolution in 2 minutes, take 0·88 cubic foot of pulp per minute, use 7 to 9 cubic feet of water per minute, and treat 14 cwt. of dry slimes per hour.

It is found difficult to construct large tables of the above type, as they become heavy to move, and are subject to jar and vibration which interferes with steady and accurate work. To get over this difficulty Linkenbach[3] invented his slime table, shewn in Figs. 263 to 265. The **Linkenbach Table** differs from the preceding in that the table is fixed, whilst the slime distributor, the clear water jets for cleaning and washing off, and the recipients for the various products, rotate; the principle of separation is identically the same as in the last case, as the dressing is performed by means of the relative motion of the above parts and of the surface of the table, so that it is indifferent which of the two is the fixed and which the rotating element. The table is

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[1] *Zeitsch. f. Berg. Hütt. u. Salin.-Wesen.* XLVII. 1899, p. 241.

[2] *Berg. u. Hütt. Ztg.* Vol. LII. 1893, p. 92.

[3] *Bull. Soc. Ind. Min.*, "Note sur le lavage des Minerais," Ch. Mouchet, Ser. 3, Vol. VIII. 1894, p. 527; Linkenbach, C., Aufbereitung der Erze, 1894, p. 527.

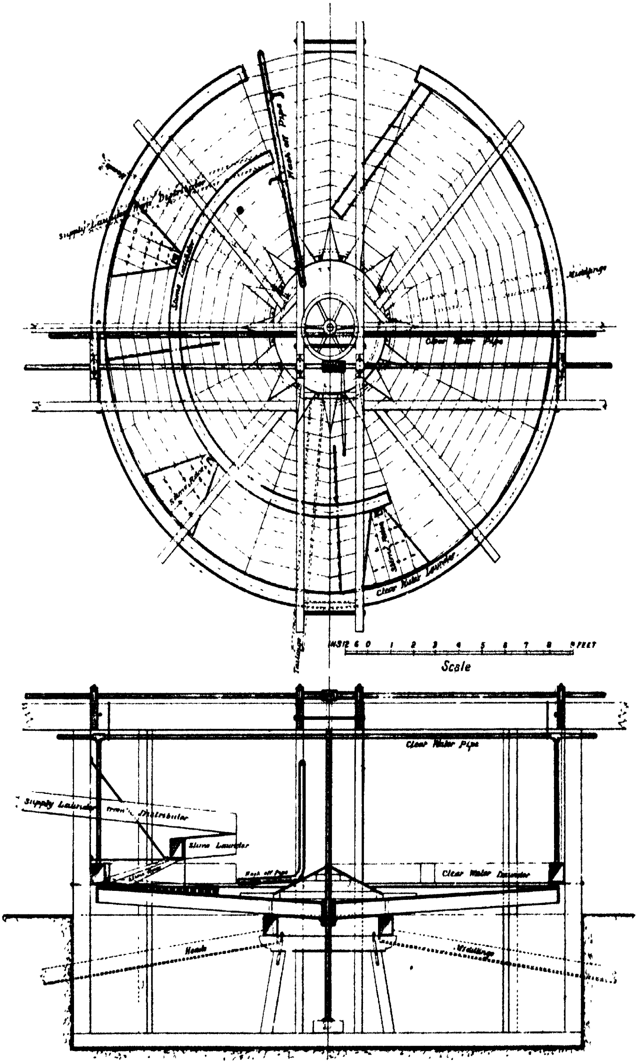

Fig. 262. Concave slime table. Plan and sectional elevation.

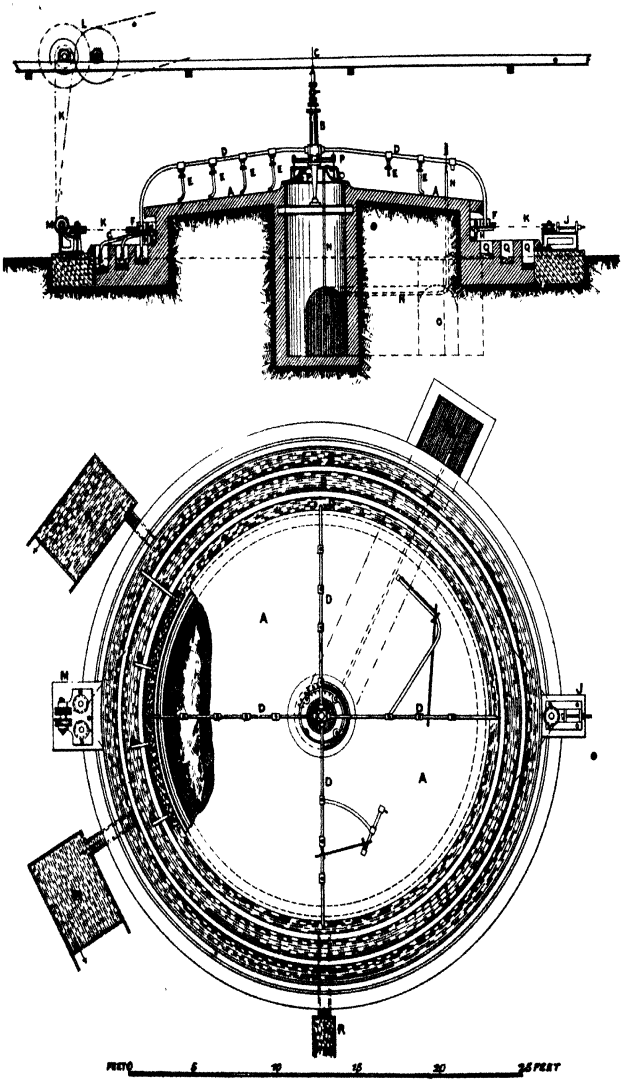

Fig. 263. Linkenbach table. Plan and elevation.

shewn in Figs. 263 and 264 in its usual forms; the latter illustration shews the details as built by the Humboldt Company. It consists of a massive conical block of concrete on an iron frame, covered with a layer of pure cement 2 inches thick, which is brought to the exact shape required by sweeping with a template, the slope given to the table *A* thus produced, being usually about ½ inch to the foot. Underneath the table a passage, *O*, is left to enable the step of the central shaft to be attended to. Around the table are three concentric cemented troughs, *Q*, each having its own separate discharge. The hollow vertical shaft *B* is fitted with a stuffing box at its upper end, through which it is supplied with clear water from a fixed main *C*. Round the lower part of this shaft is the feeding device in the form of a basin *P*, into which the pulp to be treated flows through the pipe *N*, and which distributes it uniformly over a segment of the table of about 90°. Communicating with the hollow shaft are four tubular arms *D*, from which project pipes *E* of different lengths and fitted with taps or nozzles of variable diameter as may be required; from the pipes *E* jets of water are projected onto the table for cleaning or washing off the various concentrated products. The arms *D* carry a collecting trough *F*, which is mounted on wheels or rollers *H*, running

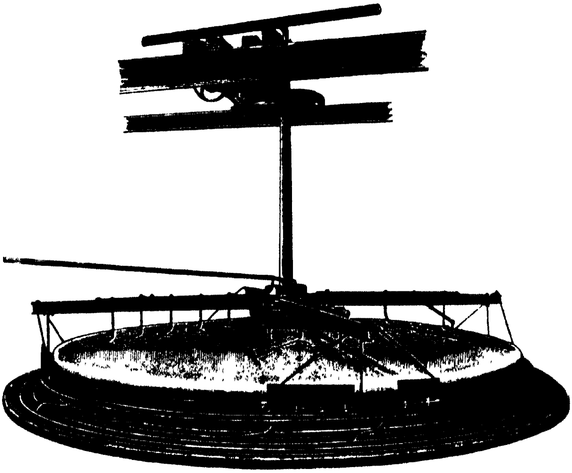

Fig. 264. Linkenbach table. Perspective.

on a circular track. This trough is divided by partitions into as many compartments as may be required, from which project discharge pipes *G* of different length, each of which delivers a separate product into one of the fixed circular troughs *Q*; each of these troughs is connected with the tanks or launders *R*, which receive separately the products collected in each trough *Q*. The driving mechanism consists of an endless chain *K* which closely grips the collecting trough *F*, and is kept tight by the tension gear *J*; at the opposite side of the table it passes over the pulleys *M* and is driven by a sprocket wheel on the counter-shaft *L*. The hollow vertical shaft, the tubular arms and the collecting trough are thus rotated together, making about 1 revolution in 4 minutes The pulp fed onto the surface of the table flows slowly down it, the lightest portion on the tailings flowing the most rapidly, and therefore reaching the collecting trough almost opposite the point of admission; the more slowly flowing middlings and concentrates will reach the trough only after the latter has turned through an angle that will be greater in proportion as the rate of flow is slower, so that those divisions of the collecting trough that are furthest away from the point opposite to the admission of the feed, will receive the heaviest concentrates. The pipes *G* discharge the tailings into the innermost trough (whence they flow off through the launder *R*) whilst the heavier constituents of the pulp are discharged into the troughs progressively more remote from the table.

There are a few modifications in this construction; the driving mechanism may consist of a worm wheel keyed to the upper end of the vertical shaft, and driven by a tangent screw, as shewn in Fig. 264; instead of a revolving gutter with tubes discharging into each fixed gutter, there may be a revolving apron of widths varying in different parts, which similarly directs the various products into their respective gutters.

A Linkenbach table working at the Vaucron[1] mines, 26 feet in diameter with its generatrix inclined at a gradient of 4·19 per cent., treats slimes containing 8 per cent. of zinc and 2 to 3 per cent. of lead; it treats 4·5 tons (dry weight) of these slimes per 8 hours, producing 2 cwt. of galena and 0·5 ton of blende; it requires about 11 gallons of clear water per minute and absorbs 2 H.P.

Such tables have been built up to 26 feet in diameter, such a table treating about 15 tons (dry weight) of fine slimes in 24 hours and an even larger amount of coarser material.

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[1] *Bull. Soc. Ind. Min.* 1894, loc. cit.

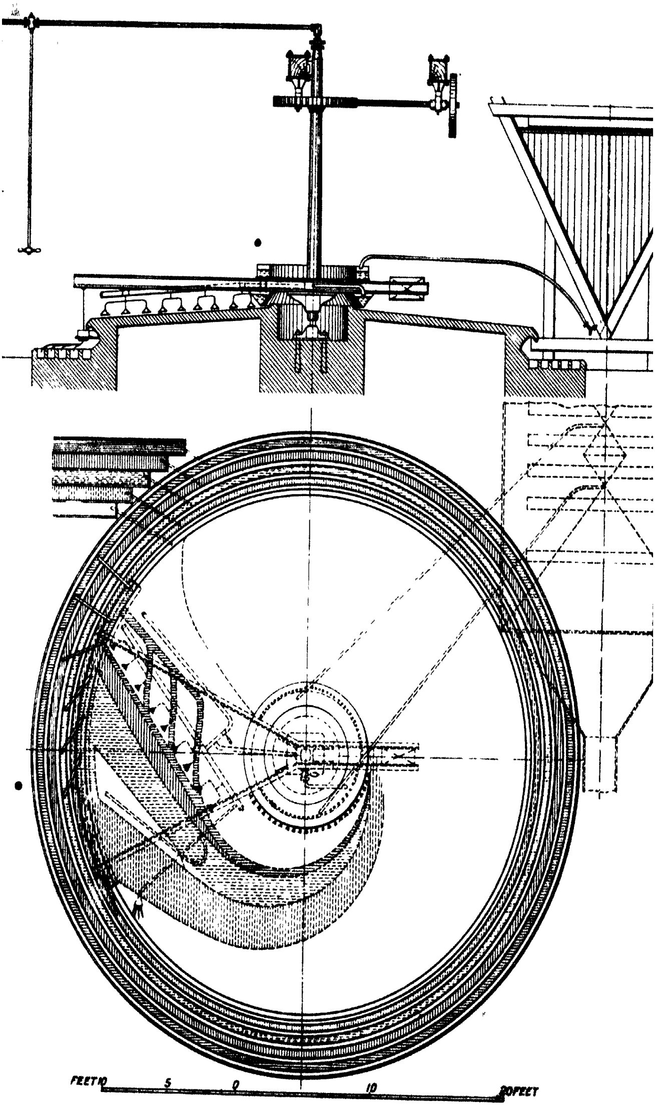

Fig. 265. Improved Linkenbach table. Plan and elevation.

According to the inventor the most suitable dimensions are:

for coarse slimes 20 feet in diameter, slope 1 in 9,
„ medium „ 21 feet 6 inches „ 1 in 10,
„ fine „ 23 feet to 26 feet „ 1 in 12.

The pulp should not contain over 8 to 10 per cent. of dry mineral; such a table can treat 4¼ cubic feet of such pulp per minute, using 3½ cubic feet of clear water for cleaning and about half as much again for washing off the concentrates. Each table absorbs 0·1 H.P.; it can treat per hour

of the coarser slimes 14·4 cwt.
„ medium „ 13·2 cwt.
„ fine „ 12·0 cwt.

producing clean concentrates, barren tailings and one or more classes of middlings.

A view of one of these tables, as built by the Humboldt Engineering Co., in a somewhat improved form is shewn in plan and section in Fig. 265, these being made of all sizes from 21 to 33 feet in diameter. Such a table 33 feet in diameter at the Maria Mine near Beuthen treats 13 tons of clayey slimes, containing 8 to 10 per cent. of zinc in the form of blende, per 10 hour shift, the product being a concentrate with 28 to 30 per cent. of zinc, whilst the escaping tailings carry 4 to 5 per cent.

These tables have also been built in several tiers, one above the other, but this arrangement can only be justified when sufficient space cannot otherwise be obtained.

CHAPTER IX.

SHAKING TABLES.

UNDER this head several machines have to be considered, which utilise a reciprocating motion for the better separation of particles of mineral conveyed in thin streams of pulp as explained in the last chapter. It must however be noted at the outset that there are two types of reciprocating motion, which though often confused under the term of "shaking" are yet quite different in their mechanical character and their effects, the first being simple shaking to and fro where the amplitude and speed of the motion in either direction are equal, and the other where there is marked difference between the two; the term shaking will here be restricted to the former type whilst the latter will be spoken of as jerking or bumping. The principles involved have been discussed on p. 226.

If a particle be placed on a horizontal surface that receives a shaking motion, the particle will remain at rest relatively to the surface until the velocity of the shake is such that the momentum of the particle exceeds the resistance due to the friction of the surface upon which it rests; the particle will then not only move with the surface, but when the direction of motion of the latter is reversed, the momentum of the former will cause it to continue to move over the surface until the effect of friction brings to rest relatively to the surface, when it will again move with it, and so on; when these conditions obtain the particle will therefore be moved to and fro upon the surface, but the motion will necessarily be equal in either direction and the particle will simply move to and fro on either side of its mean position. If a surface be inclined in a direction at right angles to the direction of its motion and pulp carrying particles of mineral be allowed to stream over it, the result of the motion will simply be to agitate the particles. It has already been shewn that particles may readily be so small as to have practically no tendency to sink in a fluid, though they would have even less tendency to rise; the result of such shaking would tend to settle them to the bottom of the layer of pulp and thus to promote their separation, but the latter would always take place in accordance with laws of separation in thin films of liquid.

If however the slope of the surface be parallel to the direction of motion, the effect is the same as in the case next to be considered, because the force required to move the particle down the slope is less than that required to move it up the slope, so that the sum of all the movements is a motion down the plane; hence this arrangement tends to move the particle down the slope even though the inclination be less than the angle of repose.

If, on the other hand, the motion of the surface is unequal in the two directions, a different effect is produced. If the motion is at different velocities, it is evident that a particle placed on the surface will be moved across the surface in the direction of the greater velocity (assuming this to be more than sufficient to overcome the friction between the particle and the surface), whilst its rate of travel will depend upon the difference between the velocities of motion in the two opposite directions. If the velocity be equal in both directions, but in one of them the swing be suddenly checked by striking against a bumping block, the motion of the surface will be stopped whilst the particle will travel onwards by virtue of its momentum, and will therefore travel toward the bumping block. In both of these cases, which are mechanically identical in their effect, the particle will travel across the surface, its rate of travelling being affected—other things being equal—by its weight, or for particles of equal volume, by their densities. This principle thus admits of the separation of particles of approximately equal size in accordance with their specific gravity, and it may therefore be used to supplement the action of separation in thin currents of pulp, as explained on p. 226. It is obvious that if a current of pulp carrying particles of mineral be allowed to stream over such a bumping surface, the heavier particles, occupying the lower strata of pulp, will be most affected by this bumping action, so that the heavier particle, even of two equal falling particles, is for both reasons moved further than the lighter one upon such a surface.

The machines that employ a true shaking motion form a group known as Vanners, the **Frue Vanner**, invented about 1874, having been the first of them; it is still amongst the best and is very largely used. It is practically a Brunton belt (see p. 324) which receives a shaking motion. It is said that the *first* attempt to combine a shaking movement with a travelling belt was made by Mr Hartwig at Moresnet in 1860[1], and that this appliance met with a certain amount of success on

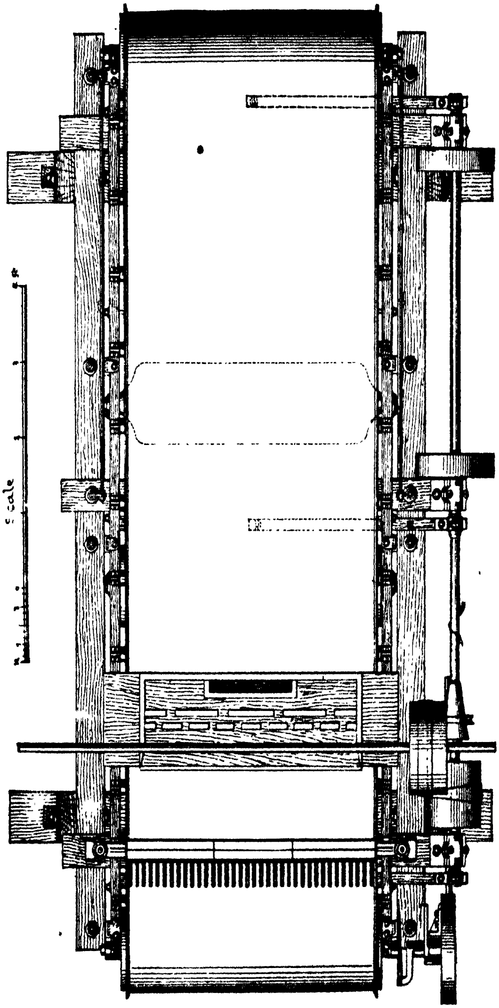

Fig. 266. Frue Vanner. Plan.

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[1] *Bull. Soc. Ind. Min.*, "Les Appareils à Toiles sans Fin, etc.," by M. F. Schmidt, 1894, p. 641.

the continent of Europe, whilst the Frue vanner was an American invention. This machine in its most modern form, as made by Messrs Fraser and Chalmers, Ltd., is shewn in Figs. 266, 266ᵃ and a perspective view of

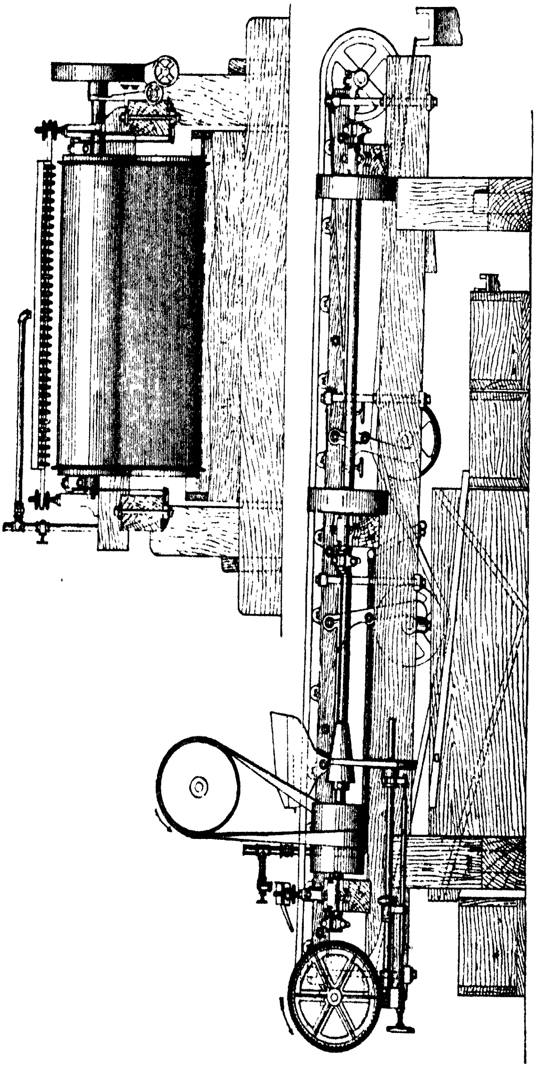

Fig. 266ᵃ. Frue Vanner. Side and end elevations.

the same machine, but mounted on iron instead of wooden standards, is shewn in Fig. 267. It consists of an endless indiarubber belt 4 feet wide with flanges at either side. This passes over a couple of drums supported in a frame about 12 feet long; the same frame carries a number of rollers over which the belt travels, thus securing a flat surface; the frame (and therefore the belt) slopes in the direction of its length, the amount of fall being from 3 to 6 inches; the belt is kept perfectly level transversely. The roller at the head (or higher) end of the machine is revolved slowly by means of a worm wheel and tangent screw, the rate of upward travel of the belt thus produced being about 6 feet per minute. By means of cranks driven off the lay shaft running along one side of the machine, the frame is kept oscillating at about 200 strokes per minute, the average length of the stroke being about 1 inch. The pulp to be concentrated is fed on to the belt from a headboard which acts as a distributor, giving a uniform flow across the whole width of the belt. About 12 inches nearer to the head end the belt receives a number of fine jets of clear water. The action of the vanner will be obvious enough from what has already been said; it is precisely that of the Brunton belt, all the particles being kept in lively motion by the vibratory action, so that none of the tailings may be entangled in the concentrates deposited on the belt. The upward motion of the heavier

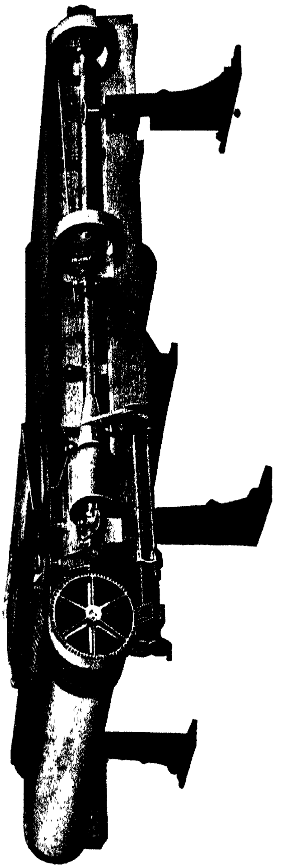

Fig. 267. Frue Vanner. Perspective.

particles is assisted by the fact that these particles adhere to the surface of the rubber belt; this is possibly due to the difference of surface tension between the water and particles of light quartzose or earthy matter and of metallic sulphides respectively. After passing the head of the machine the belt is bent down into a tank in which the adhering concentrates are washed off. The depth of pulp on the belt should be about ½ inch; it takes from 0·2 to 0·4 cubic feet of pulp per minute and about half as much clear water in addition; it thus treats about 6 tons of fine sands in 24 hours, and requires ½ I.H.P. to drive it. According to circumstances it may however treat from 5 to 8 tons per 24 hours. Each machine weighs 21 to 22 cwt. and costs £125.

Frue vanners are also built having belts 6 feet in width, which treat about 10 tons, or exceptionally up to 12 tons, in 24 hours, with a proportionally greater consumption of power. The so-called "Improved" Frue Vanner" uses a belt, the upper surface of which is corrugated; it is obvious that this modification, by hindering the flow of water down the surface of the belt, enables the belt to be driven at a somewhat greater speed, and thus increases the capacity of the machine. The plain belt is, however, usually preferred.

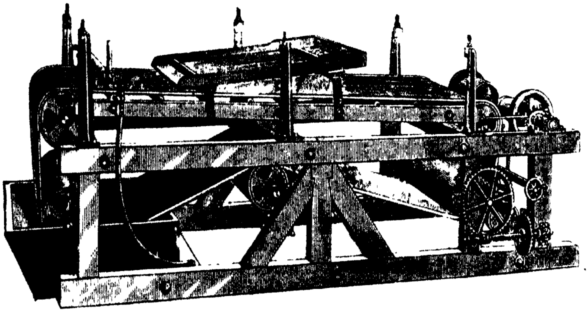

Fig. 268. Embrey Vanner. Perspective.

There are numerous other vanners, which differ but little from the Frue vanner except in matters of detail. The **Triumph** and the **Embrey Vanners** use a longitudinal instead of a transverse vibration, the mode of action being practically the same. These machines have perhaps a slightly greater capacity than the Frue vanner, but scarcely treat fine pulp as effectively. An Embrey vanner, shewn in perspective in Fig. 268 and in plan and elevations in Figs. 269, 269ᵃ with a belt 4 feet

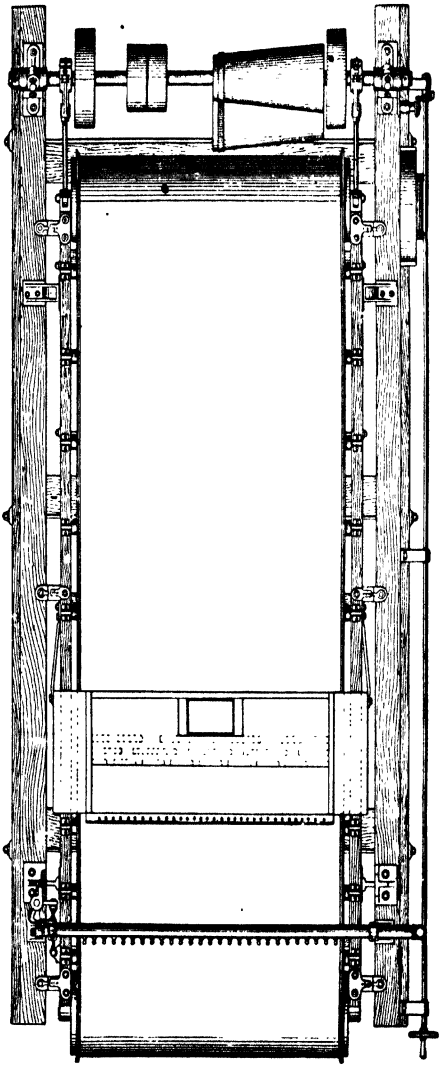

Fig. 269. Embrey Vanner. Plan.

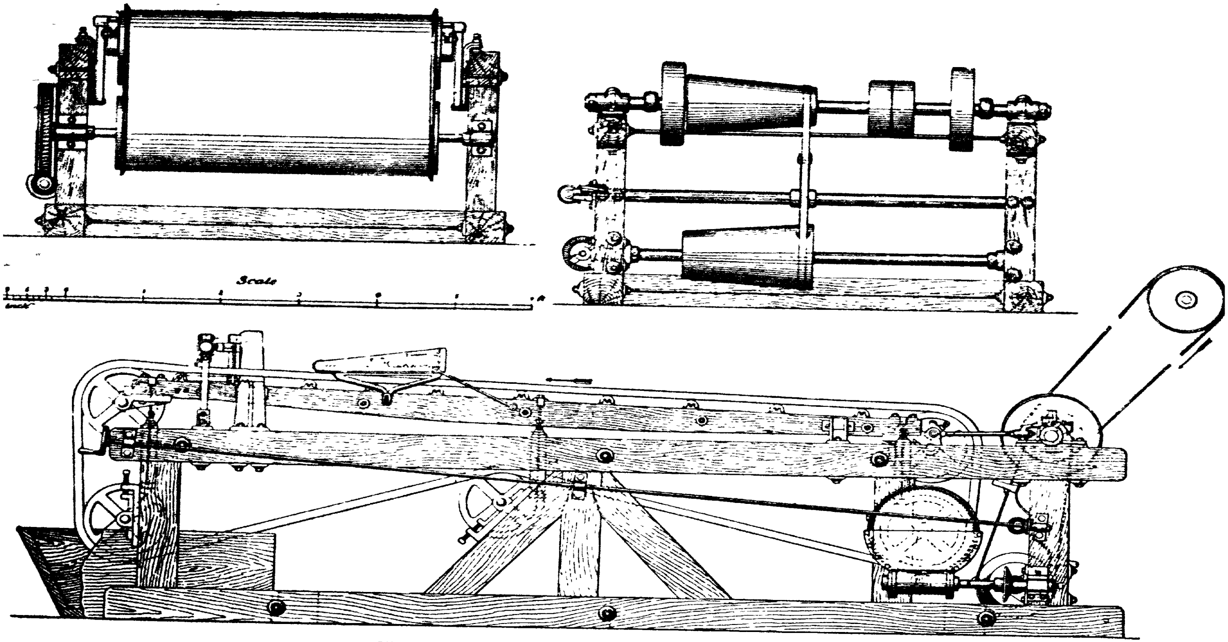

Fig. 269a. Embrey Vanner. Side and end elevations.

wide, making 220 vibrations per minute, will treat from 6 to 10 tons in 24 hours, taking 0·18 to 0·86 cubic foot of pulp per minute and about half as much clear water in addition. It is made by Messrs Fraser and Chalmers, Ltd.

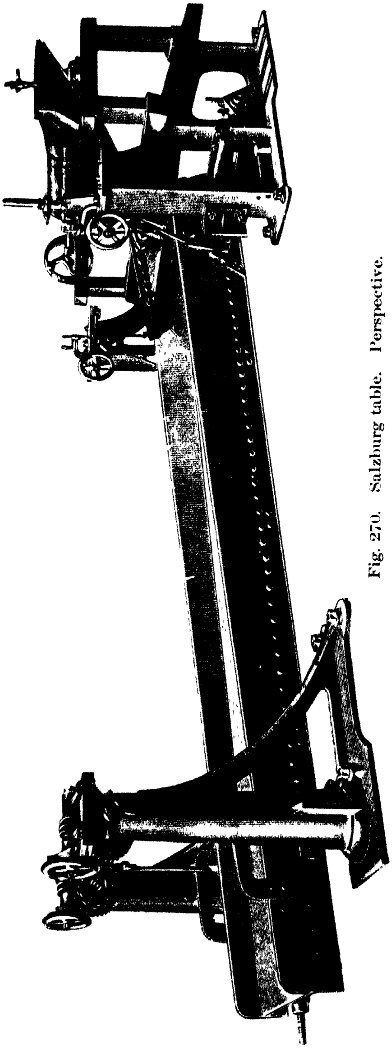

Fig. 270. Salzburg table. Perspective.

The **Woodbury Vanner** differs from the last-named in that the belt 4 feet wide is replaced by a dozen narrow belts, with the object of making the action more uniform; it has not come into extensive use.

Bumping or jerking tables fall naturally into two classes, according as the direction of the jerk is parallel or at right angles to the current of pulp. The former was the earlier construction and is well exemplified in the old **Salzburg Table,** which has been in use for a considerable period on the continent of Europe. A modernised construction of this machine as built wholly of iron by the Humboldt Engineering Co. is shewn in Fig. 270, and the more usual construction, namely a wooden table hung from iron standards, in Fig. 271[1]. It consists of a table, usually about 2½ to 3 times as

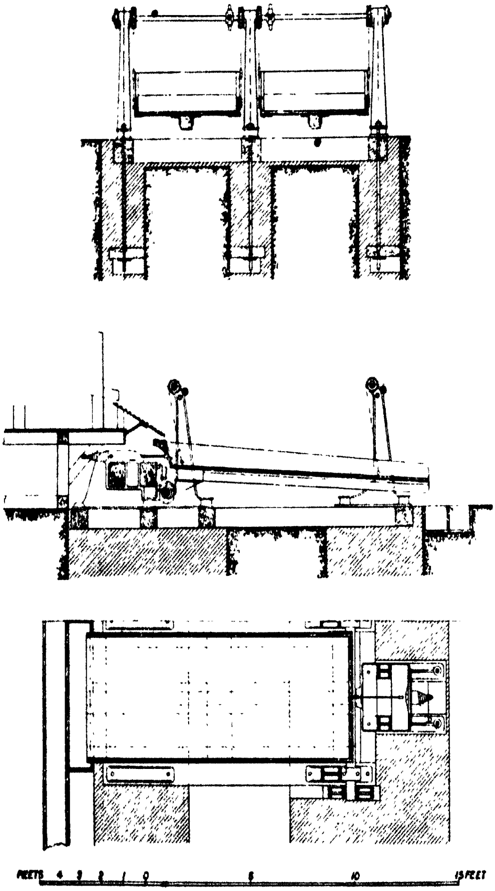

Fig. 271. Salzburg table. Plan, vertical section and end elevation.

long as it is wide, suspended by four chains in such a way as to be free to swing to and fro. The framework is of iron, the table itself being made of well-planed boards. The table is suspended from cast iron standards usually by chains or wire ropes, so as to allow the degree of inclination to be altered as required. Just below the top end of the table a three-throw cam revolves which works against a tappet consisting of a piece of stout angle iron, bolted to the lower part of the frame; the whole table and with it this angle iron are kept pressed closely against the cam by a strong spiral spring, so that the thrust of the cam takes place against the pull of the spring. The table is so suspended that the thrust of the cam lifts it slightly at the same time that it pushes it; hence as soon as the cam has cleared the tappet, the weight of the table together with the pull of the spring brings it back sharply. A stout balk of hard wood runs the full length of the table, and is securely bolted to it; the upper end of this balk is usually shod with iron, and at each swing back of the table it strikes against a massive bumping-block firmly anchored in the ground. The motion of the cam thus causes a slow movement in the direction of the slope of the table, whilst the recoil takes place sharply and rapidly ending with a bump against the bumping-block. A heavy particle placed upon the table will therefore be gradually jerked up it as the result of this motion. If pulp be allowed to run on to the table from the headboard, it is obvious that by suitably adjusting the velocity of the flow and the force of the jerks, the lighter particles can be caused to run off at the lower end, whilst the heavier concentrates will accumulate at the upper end.

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[1] Kirschner, *Grundriss der Erzaufbereitung*, Pt. II. p. 91.

The **Schemnitz** or **Hungarian Table** is practically identical with the above except for the bumping-block, which is replaced by a long elastic beam of wood. After each jerk the table is in this construction flung back by the recoil of this spring, a second bump being thus produced, and so on, each bump due to the throw of the cam being succeeded by a series of bumps—usually from five to ten—of gradually decreasing intensity, caused by this spring-board. The rate of revolution of the cam is therefore considerably slower, and the three-throw cam is often replaced by a single-armed one, but the total number of bumps, including the main jerks and the subsidiary jerks, is considerably greater. The mode of action of both forms is however practically identical.

Pulp being allowed to flow over the headboard of either form, and the table being set in motion, the result is the accumulation of the concentrated material at the head of the board, where it gradually forms a firm wedge-shaped deposit; when this has reached a depth of 6 to 8 inches the flow of pulp is stopped, and clear water is allowed to run over the table for a short time; the motion of the table is then stopped, and the deposit carefully removed with shovels. The upper portion consists of rich heads, which require re-treatment upon a similar table to produce clean concentrates; the next section is of about the same composition as the material to be treated and may be returned for re-treatment upon the same table, whilst the lower section is poor and is usually further treated upon another similar table. As a general rule these tables are worked in pairs.

With a table from 5 to 7 feet wide, from 2 to 3 cwt. of dry material can be treated per hour on the average, the power consumption increasing from ½ H.P. when the table is empty to 1 H.P. when it is filled.

Some of the chief data concerning these tables are as follows:

	For slimes	For fine sands
Inclination of the table	2	5
No. of blows per minute (Salzburg table)...	100	80
Length of throw (Salzburg table)	1¼″	2½″
No. of direct blows per minute (Schemnitz table)	18	12
No. of secondary jerks following each main blow (Schemnitz table)	5–6	10
Length of throw (Schemnitz table) .. .	½″	3″
Time required to fill table with pulp carrying 5% to 10% of concentrates	8–9 hours	2–2½ hours
Quantity of pulp per minute per 5 foot table	¾–1 gallon	3½–5 gallons
Dry material per gallon of pulp........	⅞–1¼ lbs.	3½–7 lbs.

These tables were very largely used in Western Germany and Austro-Hungary, chiefly for the treatment of lead ores, and are still in use to some extent; they are, however, being rapidly displaced by modern continuous-acting machines, their small working capacity, and the large amount of labour required (usually given as three men per pair of tables per shift) being grave drawbacks, whilst the degree of concentration attained is by no means correspondingly satisfactory.

A very similar table, shewn in Fig. 272[1], has been used a good deal in Australia, where it was generally known as the **Halley Table.** In this, as in those last described, the concentrates accumulate at the head of the table, whence they are removed with a shovel, whilst the barren tailings flow off continuously at the foot. Such a table, 4 feet wide by 8 feet long, working at 150 to 200 blows with 1½ to 2 inch throw per minute will treat about ½ ton of pulp per hour, this pulp being usually that produced by crushing pyritous gold quartz in stamp mills.

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[1] Louis, *Handbook of Gold Milling*, 3rd Ed. p. 359.

This type of table has undergone considerable improvement in the Western States of America by making it automatic and continuous-acting; this has been done by simply adjusting the working conditions so that the heavier concentrates are discharged continuously over the head of the table, whilst the lighter, practically barren tailings, flow off at the foot. Such a table is that known as the **Gilpin County, Gilt Edge,** or **Golden Gate Concentrator,** all of these being practically the same machine. A form of this as made by the Colorado Iron Works Co. of Denver is shewn in isometrical projection in Fig. 273, and the details of it in Fig. 273ª. The same firm's "Perfection," Gilpin County

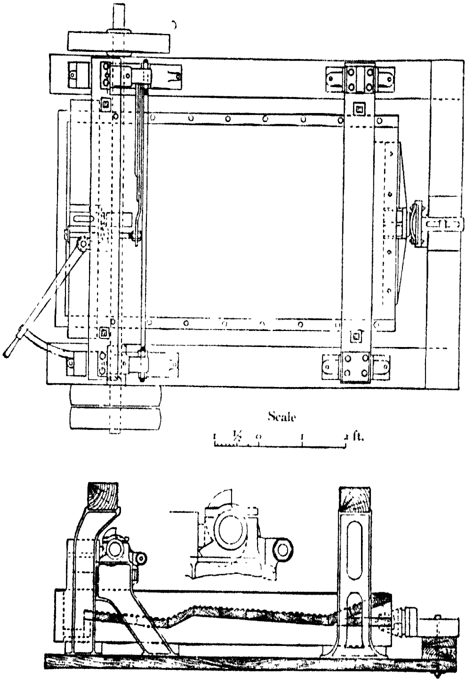

Fig. 272. Halley Table. Plan and sectional elevation.

Bumping Table, is shewn in Fig. 274; its construction and mode of action are practically identical with those of the last-named.

All these machines have the advantage of fair efficiency, and at the same time their cost is moderate.

An ordinary Gilpin County concentrator will treat from 10 to 15 tons of ordinary sands per 24 hours.

Machines of this type have been used not only upon fine sands and slimes, they have also been employed to some extent for dressing coal, and two of these may be mentioned, namely the Campbell and the Craig coal washers.

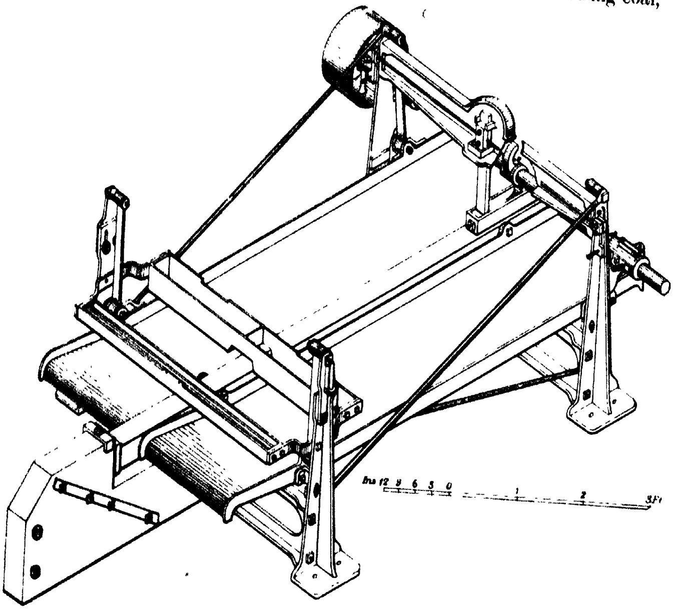

Fig. 273. Gilpin county concentrator. Isometric projection.

The **Campbell Coal Washer**[1] is shewn in Fig. 275; it consists of a table 9 feet by 2 feet 6 inches, suspended by 4 iron rods, of which the pair nearest the lower end of the table can be raised and lowered, so as to alter the slope of the table as required. The surface of the table is composed of a series of steel strips, forming transverse riffles, which

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[1] *Trans. Inst. Min. Eng.* Vol. XXIII. p. 435.

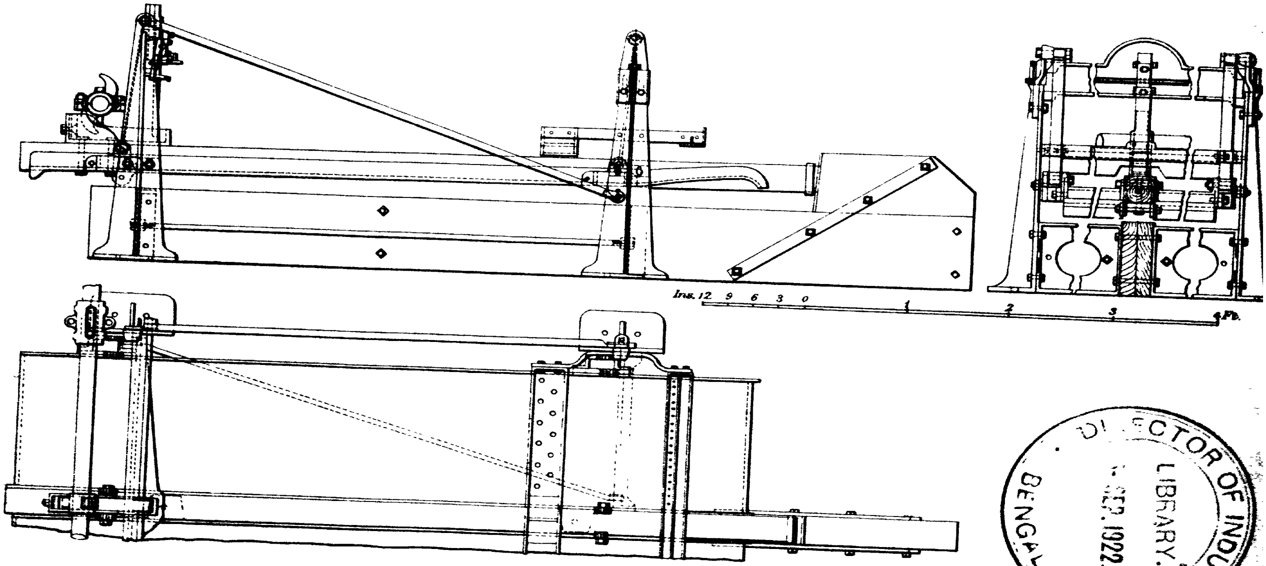

Fig. 273a. Gilpin county concentrator. Plan, side and end elevations.

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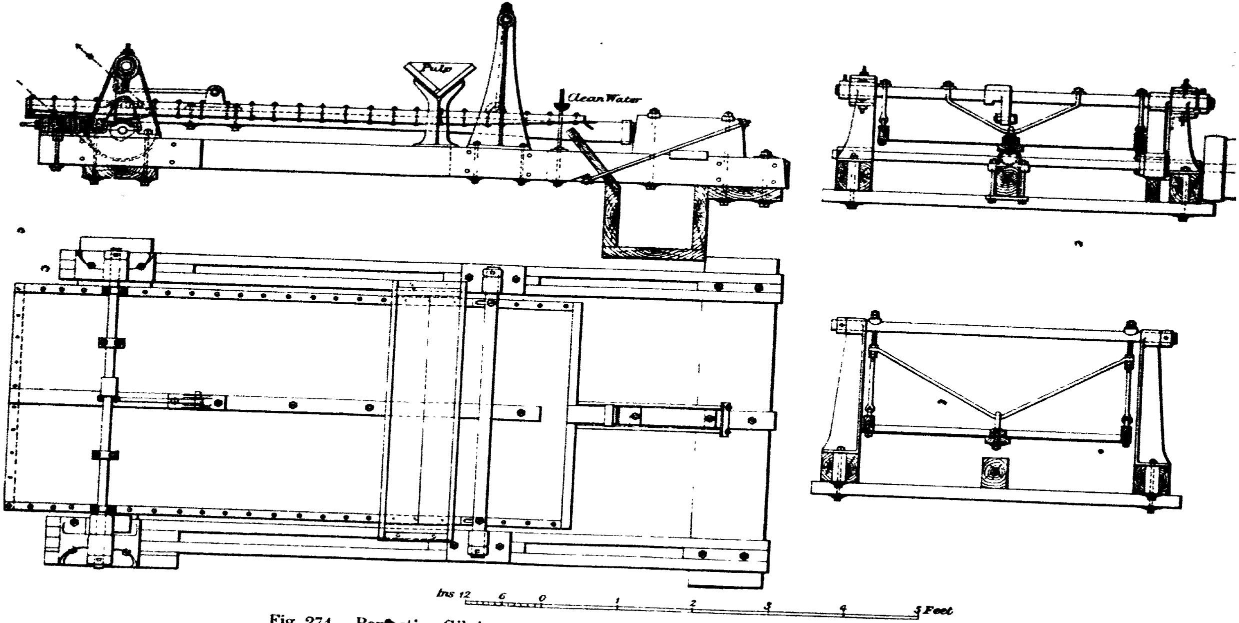

Fig. 274. Perfection Gilpin county bumping table. Plan, side and end elevations.

are separated from the true bottom of the table by a space of 1½ inches. The table is set in motion by a cam and lever, and along the bottom and

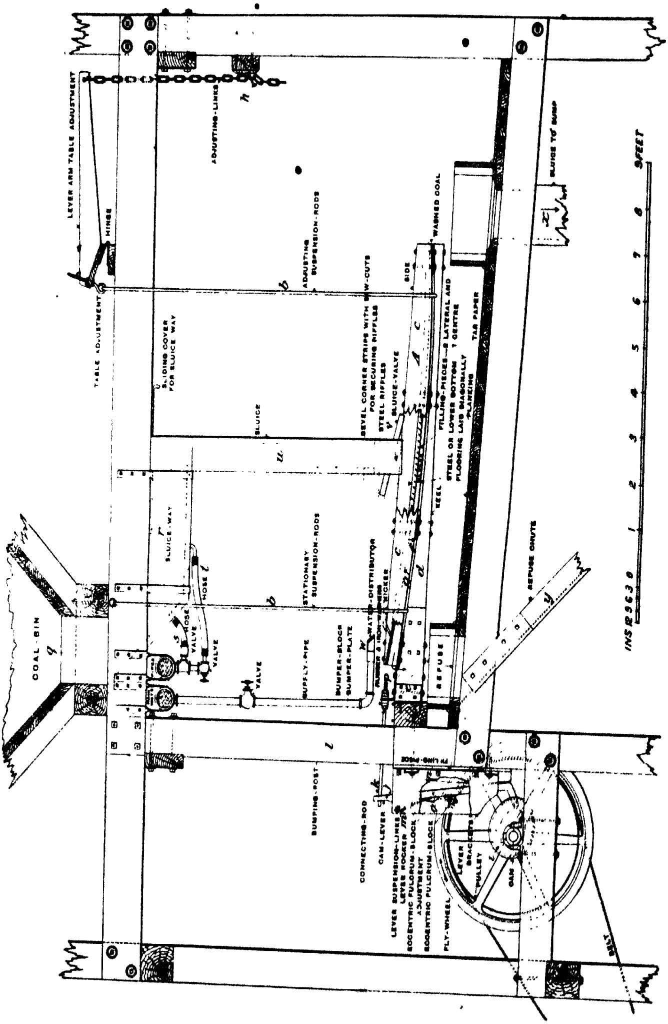

Fig. 275. Campbell coal washing table.

underneath it runs a stout oaken beam that strikes against a massive wooden bumping-block. The cam is so arranged that the table is moved slowly forwards (in the direction in which it slopes) and rapidly in the opposite direction ending with a sharp bump.

The coal to be washed is sluiced on to the middle of the table from a bin, and a current of clear water runs on near the head of the table. The result of this arrangement is that the cleaned coal is delivered at the lower end of the table over the surface of the riffles, whilst the coarser dirt is similarly discharged at the head end, the finer dirt passing through the riffles and being discharged over the true bottom of the table, also at the head end. It is stated that this table works well on unsized coal under 1 inch mesh; that its capacity is 5 to 7 tons per hour with a consumption of ½ H.P., and a water consumption equal to 250 gallons per ton of coal. The sulphur and ash in the raw coal are said to have been 1·910 per cent. and 5·810 per cent. respectively, and

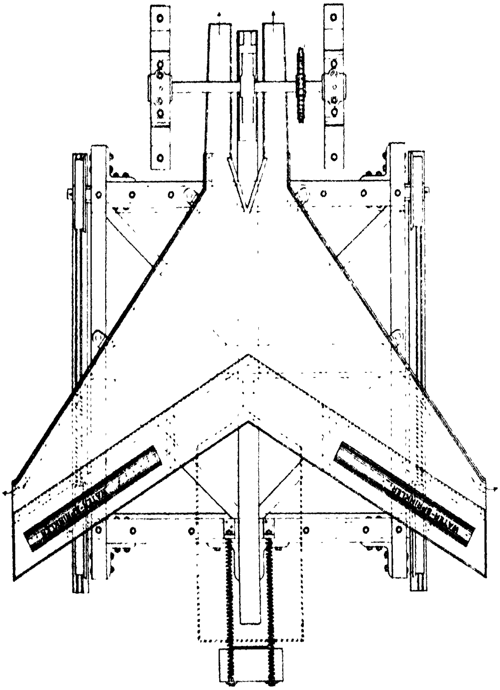

Fig. 276. Craig coal washing table. Plan.

in the washed coal 0·857 per cent. and 4·800 per cent., these results having been obtained in Pennsylvania on Vinton coal.

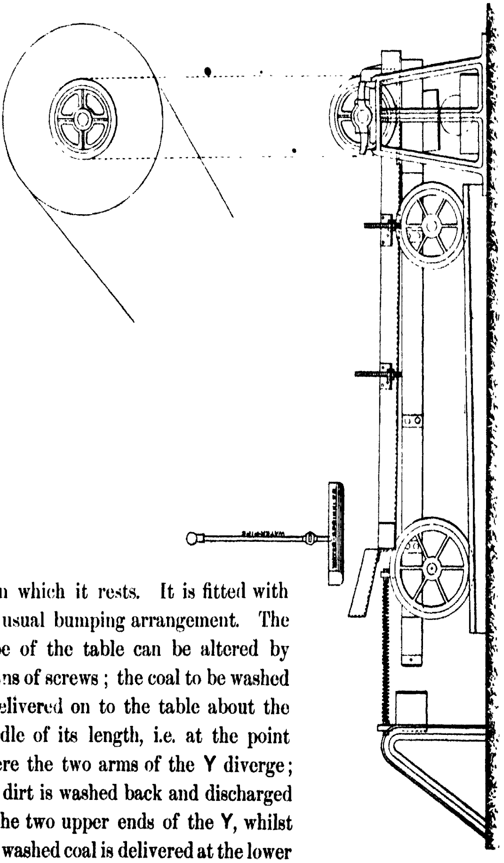

Fig. 276^{a}. Craig coal washing table. Elevation.

The **Craig Table**, Figs. 276 and 276^{a}[1], consists of a table Y-shaped in plan, running upon wheels, actuated by a two-armed cam, which works against a strong spring, forcing the table up the gently inclined track upon which it rests. It is fitted with the usual bumping arrangement. The slope of the table can be altered by means of screws; the coal to be washed is delivered on to the table about the middle of its length, i.e. at the point where the two arms of the Y diverge; the dirt is washed back and discharged at the two upper ends of the Y, whilst the washed coal is delivered at the lower end, which forms a pair of narrow spouts. The table will wash coal up to 1½ inch mesh; working at 60 five-inch strokes per minute, it has been found capable of washing about 8 tons of coal per hour, and at Coanwood Colliery, where it was tried upon coal containing 11·51 per cent. of ash, and 1·96 per cent. of sulphur, these figures were reduced to 4·80 per cent. and 1·657 per cent. respectively in the washed coal.

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[1] *Trans. Inst. Min. Eng.* Vol. XXIII. p. 179.

The first of the bumping tables to use a transverse motion was the **Rittinger Continuous Table,** the invention of which about the middle of the 19th century marked a great advance in concentrating machinery. In its original form it consisted generally of two tables side by side made of wood upon a substantial wooden frame, each table being about 8 feet long by 4 feet wide, suspended by four ½-inch iron rods. The table was pushed by a cam against a strong wooden spring, which jerked it back sharply against a substantial bumping-block. The result of this motion, as already explained, is to move a particle lying on the table across it in the direction towards the bumping-block. A headboard about 1 foot wide feeds a stream of pulp on to the top of the table at the side furthest from the bumping-block, whilst a stream of clear water flows over the remainder of the table. The arrangement is shewn diagrammatically in plan in Fig. 277, where *A* represents the bumping-block, *B* the supply of pulp and *C* the supply of clear water, the arrow shewing the direction in which the table is jerked by the spring. It is obvious that a mineral particle discharged at *B* upon the surface of the table is carried down the table by the flow of pulp at a continuously increasing velocity, whilst it is moved across the table towards *A* by a series of equal impulses. A heavier particle will be carried more slowly down the table than a lighter one, and at the same time its momentum carrying it towards *A* will be greater, so that of two particles the heavier will move further across the table before it reaches the lower edge; all particles will move in parabolic curves, the lightest being discharged at some point such as *a*, the heaviest at a point such as *b*, and particles of intermediate specific gravities at points between the two. By inserting suitable divisions, any number of products of different specific gravities may be collected as required.

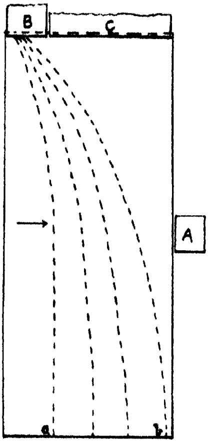

Fig. 277. Diagram of action of Rittinger table.

The table in one of its original forms is shewn in plan and in section on the line *AB* in Fig. 278, and in longitudinal section on *CD* in Fig. 278*, whilst Fig. 279 is a perspective view of it as made by the Humboldt

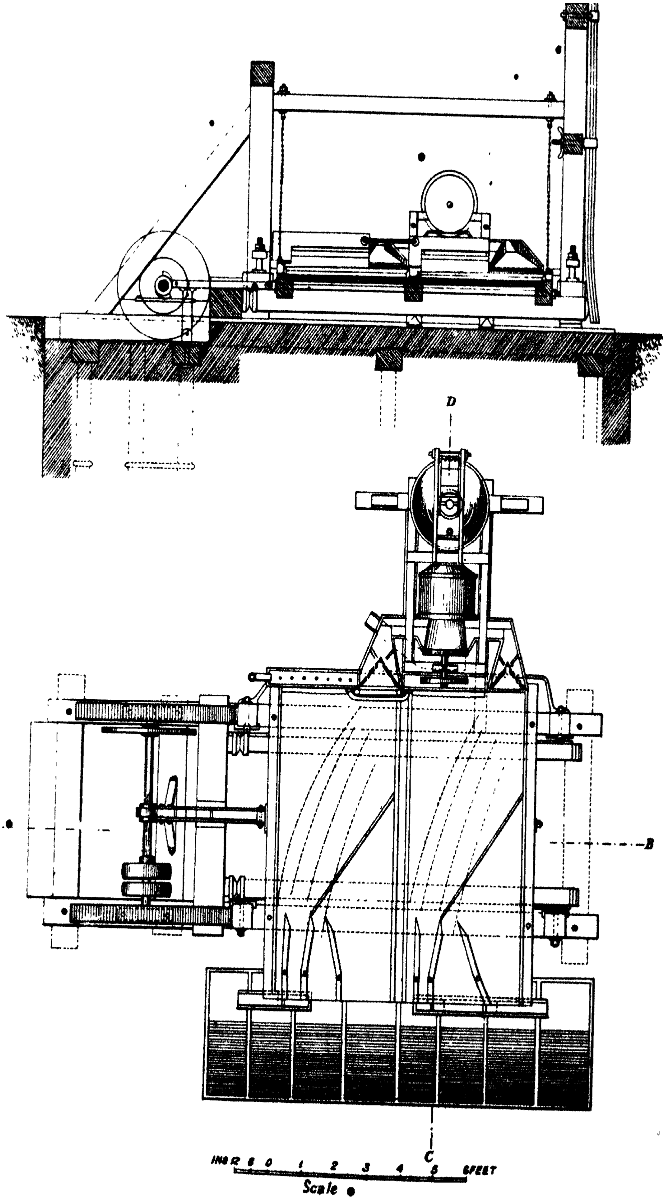

Fig. 278. Rittinger table. Plan and cross-section

Engineering Co. in a more modern form, in which the wooden standards, cam, spring, etc. are replaced by iron parts, thus making a more compact as well as a stronger machine. These tables are still usually made about 8 feet long by 4 feet broad, of which 1 foot is taken up by the pulp head-board and the remainder by the stream of clear water. The quantity of pulp should be from 0·1 to 0·15 cubic foot per minute for slimes and about twice as much for sands; the consumption of clear water in the former case is about $\frac{3}{4}$ cubic foot per minute, but may be considerably increased when coarser material is treated. The quantity of dry material that can be treated ranges from about $\frac{1}{2}$ cwt. of slimes to about 2 cwt. of sands per hour. According to the inventor the surface should have a slope of from 3 to 6 degrees, and the number of blows should vary from 70 to 100 per minute. The power consumption of a pair of such tables is about $\frac{1}{4}$ H.P., and one man can readily attend to two such tables. These machines may still be found in occasional use, but have generally speaking been replaced by others employing the same basal principle but having greater working capacity.

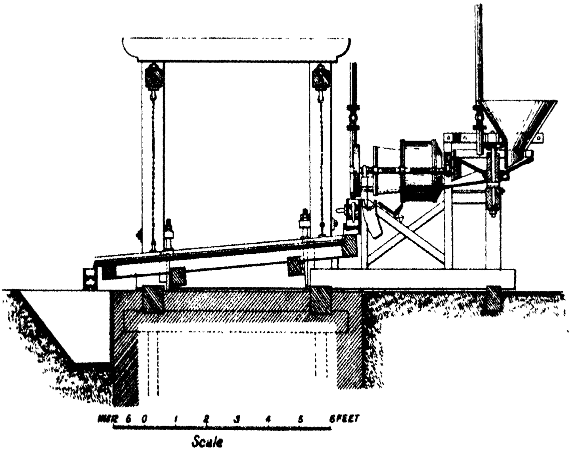

Fig. 278a. Rittinger table. Sectional side elevation.

At Diepenlinchen[1] Rittinger tables with glass surfaces have been used; they make 260 strokes per minute and treat 120 lbs. of material per hour.

Kavan of Przibram[1] has modified the Rittinger table by replacing

[1] *Berg. u. Hütt. Ztg.* Vol. LV. 1896, p. 13.

the latter, 5 feet wide and 8 feet long, by two narrow tables 2 feet 3 inches wide and 3 feet 3 inches long side by side, three such double tables being arranged, one below the other, with 8 inch drops between them. This composite table is known as the **Kavan Repeating Table**;

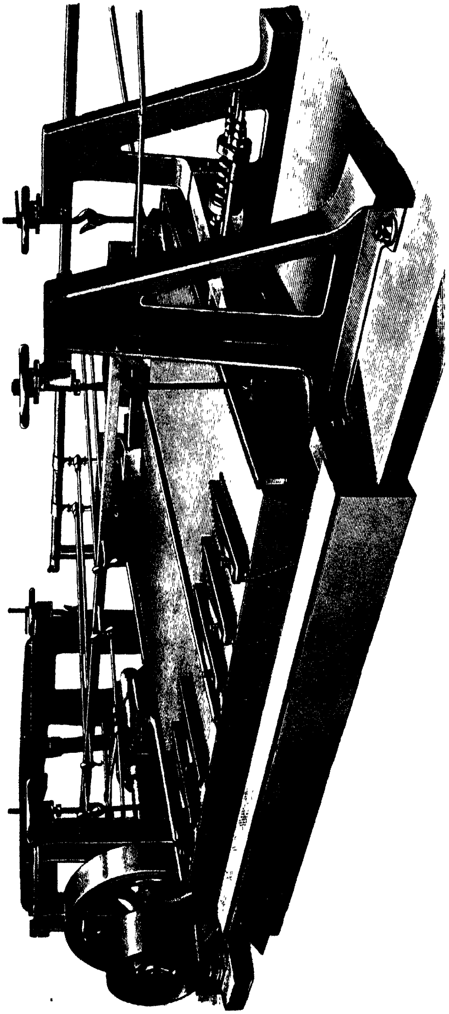

Fig. 279. Modern Rittinger table. Perspective.

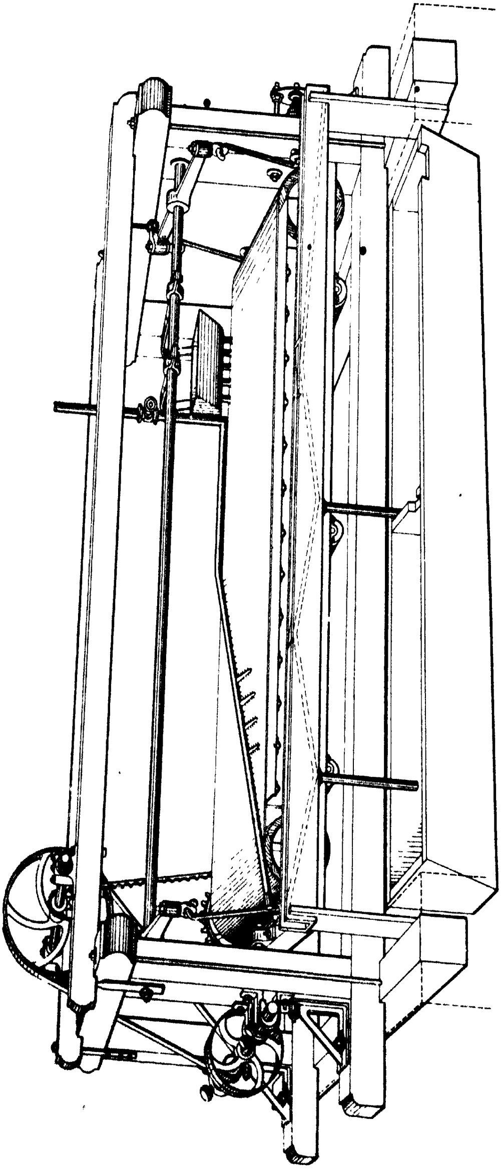

Fig. 280. Lührig Vanner. Perspective.

it is claimed for it that it is much lighter than the older form and can be run at 280 to 300 blows per minute as against 100; it will treat 4 cwt. of fine sands per hour with a flow of pulp equal to 2·1 cubic feet per minute and an equal supply of clear water; the power consumption is ⅓ to ½ H.P.

The **Lührig Vanner,** which is practically identical with the **Bilharz-Stein** continuous shaking table, is shewn in Fig. 280, and the Bilharz-Stein Table in Fig. 281, this being the form made by the Humboldt Engineering Co. It practically consists of a Rittinger continuous shaking table, the upper surface of which is covered by an endless indiarubber belt, which has a slow motion towards the bumping block side, so that the motion of the belt supplements that due to the bumping action, and thus assists the effect of the latter in separating the heavier from the lighter constituents.

As shewn in the Figure, it consists of a substantial wooden or iron frame, from which is suspended a light frame of wrought iron, carrying a drum at either end, over which the rubber belt is stretched. The suspended frame can be adjusted at any desired angle of slope by means of the arrangement shewn in the figures. The frame receives a series of jerks by means of a cam or a crank at the motion end, and the belt is at the same time caused to travel slowly by the revolution of one of the drums. The belt is supported at intermediate points by rollers in the Lührig table, whilst in the Bilharz-Stein table it rests upon wooden boards, with grooves cut into their upper surfaces, down which water is allowed to flow so as practically to keep the belt floating and allow it to move with very little friction. The pulp is delivered over a headboard close to the motion end, clear water to wash the heavier portions left on the belt, and ultimately to wash them off the belt, being supplied by the diagonal pipe shewn, which is suitably perforated. Along the lower edge of the belt are a number of receptacles or usually one long trough divided by partitions into a number of compartments, through which clean concentrates, middlings (often 2 or 3 grades) and waste are allowed to flow off. The great advantage of this machine is that it enables as many different grades of product to be obtained as may be desired.

The Lührig vanner has usually a belt about 12 ft. long by 3 ft. 6 ins. wide, travelling at 8 to 10 ft. per minute, and receiving about 180 bumps per minute, the length of stroke being ⅛ to ¾ inch. It will treat from 3 to 8 tons per 24 hours, with a power consumption of ⅓ to ½ H.P. and a water consumption of 5 to 10 gallons of clear water per minute. The

Bilharz-Stein table has a 3 ft. belt and is worked at about 160 impulses per minute. Its capacity is from 2½ to 7½ tons per 24 hours with a power consumption of about ⅓ H.P. and a water consumption of 7 to 11 gallons per minute. At Mühlenbach[1] these tables work at 200 strokes

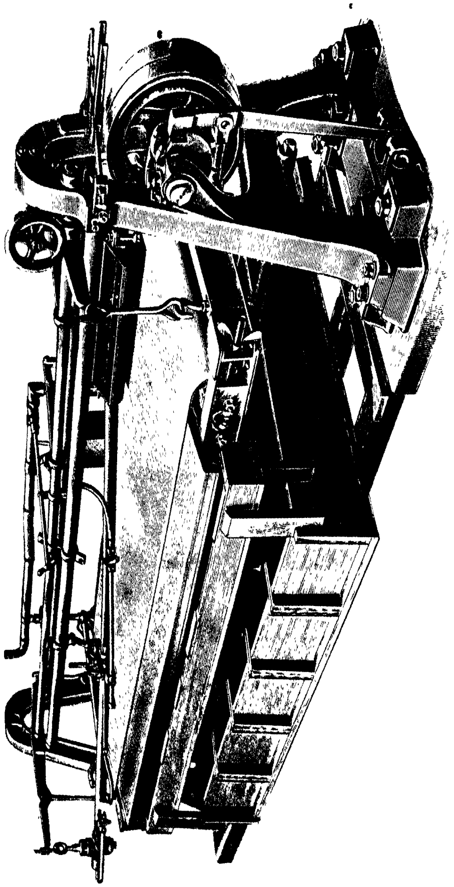

Fig. 281 Bilharz-Stein table. Perspective.

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[1] *Berg. u. Hütt. Ztg.* Vol. LV. 1896, p. 15.

per minute, the rate of travel of the belt being 11 ft. per minute; it treats 4¼ cwt. per hour with a clear water consumption of about 12 gallons per minute.

The **Wilfley Slime Table,** Fig. 282, is a somewhat similar machine, except that the continuous belt is replaced by a number of separate shallow trays or compartments, the bottom of each of which is covered with canvas, linked together so as to form a belt, the long axis of the tray lying across the width of the belt and therefore at right angles to its direction of motion. Each tray makes a complete circuit of the machine in about half an hour and receives about 180 impulses per minute. The gradient of the trays and the flow of water are so adjusted that the lighter worthless portions flow off the trays, whilst the heavier slimes remain on the canvas surface and are washed off as these pass to the underside of the machine, which thus makes only two grades, namely, concentrates and tailings. Its capacity is said to be 15 tons in 24 hours, and its power consumption 1 H.P. This machine is comparatively novel and has not yet been introduced into general use.

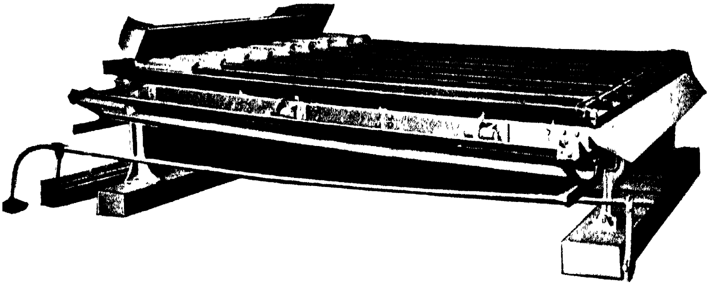

Fig. 282. Wilfley slime table. Perspective.

An important group of machines has come into extensive use within recent years, the precursor of all of which has been the **Wilfley table**, this being the first to introduce certain novel principles. This table consists essentially of a Rittinger table, the upper surface of which is covered with a series of grooves or riffles at right angles to the direction of flow of the pulp, these grooves being deepest at the end at which the pulp is admitted and gradually running out to nothing at the further end. The motion is not a true bump, but its equivalent, produced either as in the Wilfley table by a quick forward stroke and slow return,

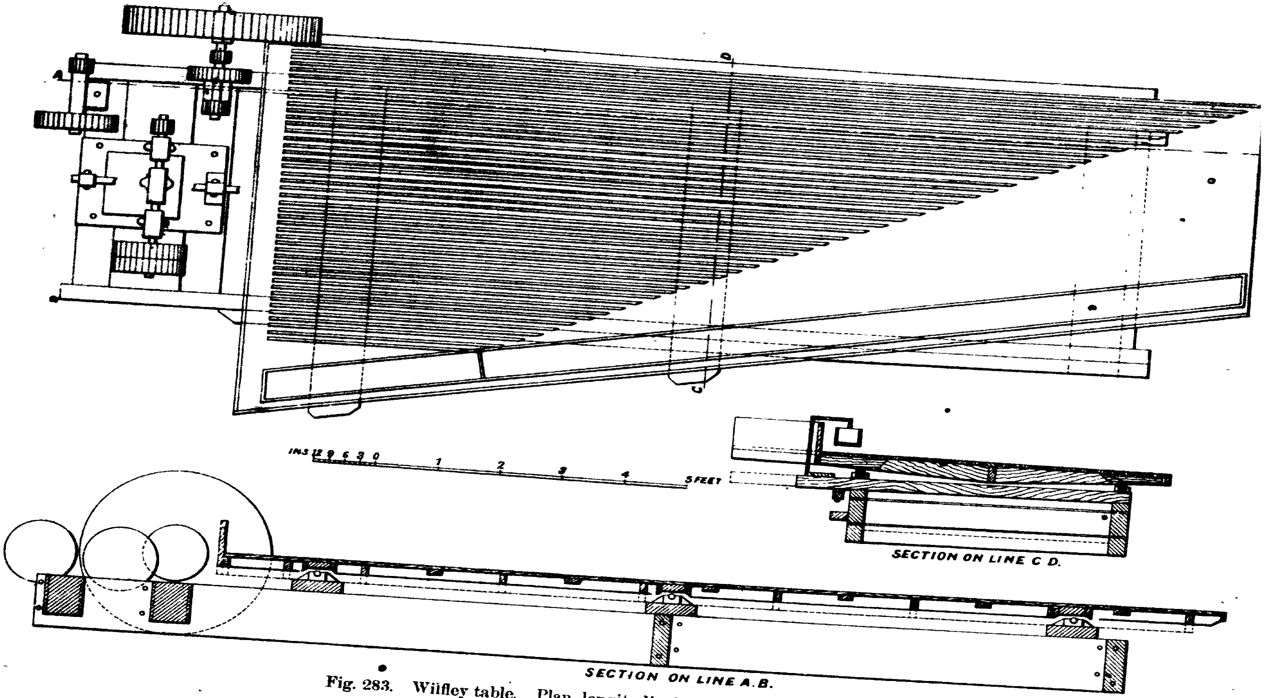

Fig. 283. Wilfley table. Plan, longitudinal and transverse sections.

or as in the Ferraris table by a throw upwards in one direction and a movement downwards in the other.

The Wilfley table is shewn in plan and section in Fig. 283 and in perspective in Fig. 284. It consists of a trapezoidal table, the shape of which is indicated in Fig. 283, being somewhat narrower at the upper than at the lower end, and about 16 ft. long by 7 ft. maximum width, its length being not as in the Rittinger table in the direction of flow of the pulp, but at right angles to it. The table is supported so as to be capable of moving freely, and is moved by a link motion, which gives it a quick throw forward—i.e. away from the motion end—and a slow movement backwards. The table is covered with linoleum and slopes upwards from the motion end, the total slope being about ½ inch; on the table are nailed a series of strips of wood, about ⅜ inch wide which gradually increase in length from the upper to the lower edge of the table, occupying nearly the full length at the lower edge. These strips are ⅜ inch deep at the motion end and taper out to a feather edge at the discharge end, so that the riffles formed by them gradually decrease in depth from ⅜ inch to nothing, the upper surfaces of the wooden strips being nearly horizontal. The table also has a slope, adjustable at will, from the back to the front edge. The pulp is delivered over a headboard about 3 feet wide, close to the motion end, the rest of the table receiving a supply of clear water. Assuming the table to be in motion, the pulp flowing down fills successively the deep ends of the riffles; in each riffle separation takes place, the heavier particles sinking

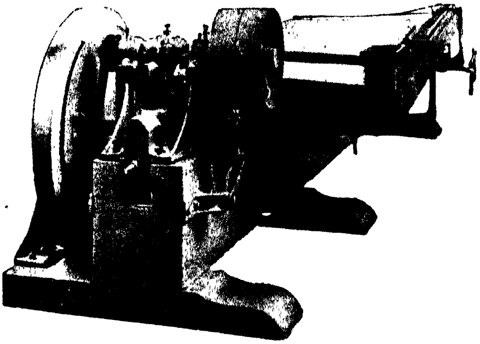

Fig. 284. Wilfley table. Perspective.

to the bottom of the riffle and being gradually propelled along it, whilst the lighter particles overflow from riffle to riffle until they flow off at the lower edge, the pulp having thus to undergo concentration in each riffle, so that the escaping waste tailings cannot easily carry off entangled particles of heavier materials. The tailings are therefore usually clean enough to be allowed to run at once to waste. The heavier materials accumulate in the riffles, chiefly of course in the upper ones, and are caused to travel along these by the jerking action of the table, until they have reached a portion of the riffle so shallow that they can be carried over by the stream of clear water into the next lower, and so on. The ultimate result is that the heaviest particles are discharged from the riffles at their ends, whilst the lightest run straight across the table, intermediate products being obtained at intermediate points. A certain proportion of middlings is collected by a raff wheel, and returned by it to the headboard, although the utility of this arrangement is open to question. This arrangement of riffles, combined with the great length of the table in the direction of mechanical motion thus accentuates the effect of the Rittinger table and causes the Wilfley table to be a far more efficient machine, separating the different classes over a greater distance. It also allows of a greater speed of working, and thus produces a machine of large capacity, which, when once adjusted, requires very little attention. It can treat comparatively roughly sized stuff, but is not suited to fine slimes; it can treat particles up to ⅛ inch in diameter quite successfully. The machine makes on an average 240 three-quarter inch strokes per minute, and requires rather over 1 H.P. to run it and a supply of water varying from 5 to 20 gallons per minute according to the nature of the pulp treated. A table will treat on an average 30 tons per 24 hours, but has been known to deal satisfactorily with more, up to 50 tons having been treated on one table. The machine weighs 22 cwt. and costs about £90.

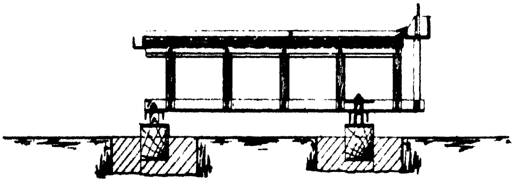

Fig. 285. Buss table. End elevation.

A table working on similar principles is the **Buss Table, or Lührig**

Vanning Table. It is shewn in Figs. 285 and 285ᵃ. The table is moved by a simple eccentric, but as it is supported on springs which slope

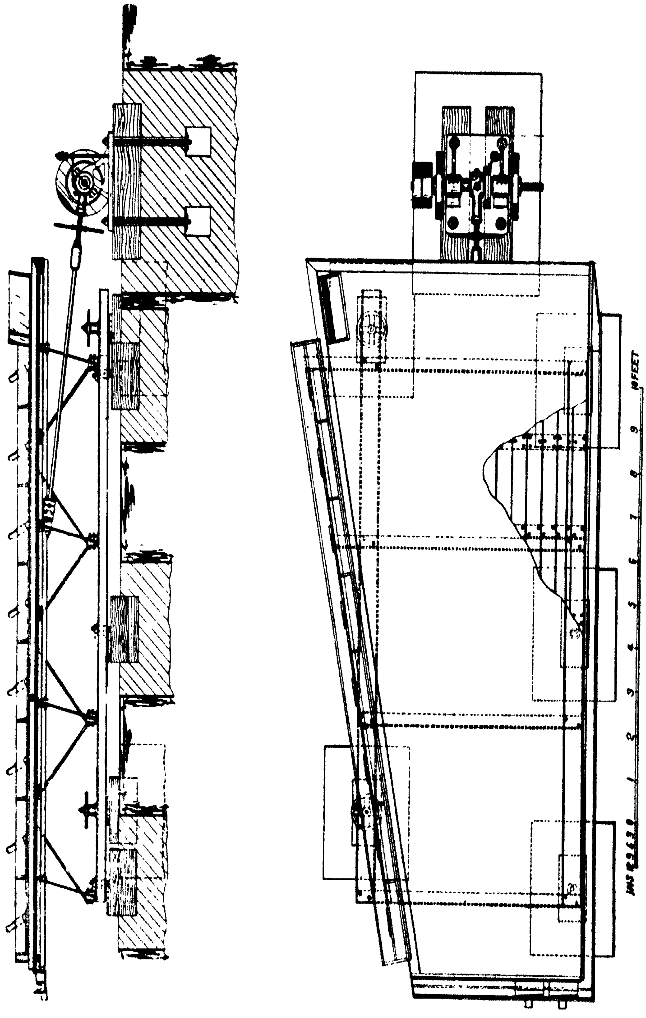

Fig. 285ᵃ. Buss table. Plan and side elevation.

towards the motion end, an upward and forward throw is given to the material resting upon it. The surface of the table is covered with linoleum; the patentees state that it may either be used plain (like the Rittinger table) or provided with riffles (like the Wilfley table), the latter being the more usual arrangement. The riffles are set obliquely across the table sloping downwards from the motion end, for about $\frac{3}{4}$ of their length, and then slope still more rapidly towards the lower edge of the table, being carried out quite to the edge. The surface of the table is 16 feet long, and it is 7 feet 6 inches wide at the motion end and 4 feet 6 inches at the discharge end. It is supported upon 16 springs made of ash pivoted at their lower ends, and the stroke of the eccentric is variable. The speed at which the table is run is 250 to 290 strokes per minute; its power consumption is given as $\frac{1}{2}$ to $\frac{3}{4}$ H.P. and its water consumption as 6 to 9 gallons per minute. Its weight is $2\frac{1}{2}$ tons and its price is £80. It is said to be capable of treating 12 to 36 tons per 24 hours. From data published by Mr Dietzsch of the working of eight such tables at the Clitters United Mines[1], Cornwall, it would seem that each table there treated about $8\frac{1}{2}$ tons of ore per 24 hours.

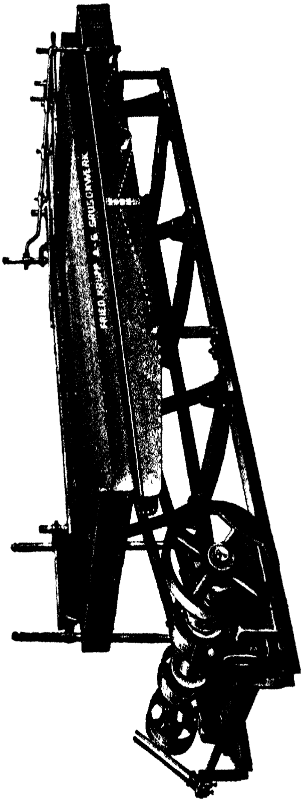

Fig. 286. Ferraris table. Perspective.

The **Ferraris Table,** Fig. 286, is very like the last, being supported and driven in the same way; the surface is also covered with linoleum and supplied with longitudinal riffles. It is built by several makers, among the best known being the Krupp Grusonwerk Company; they make two

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[1] *Trans. Inst. Min. Met.* Vol. xv. 1905–6, p. 2.

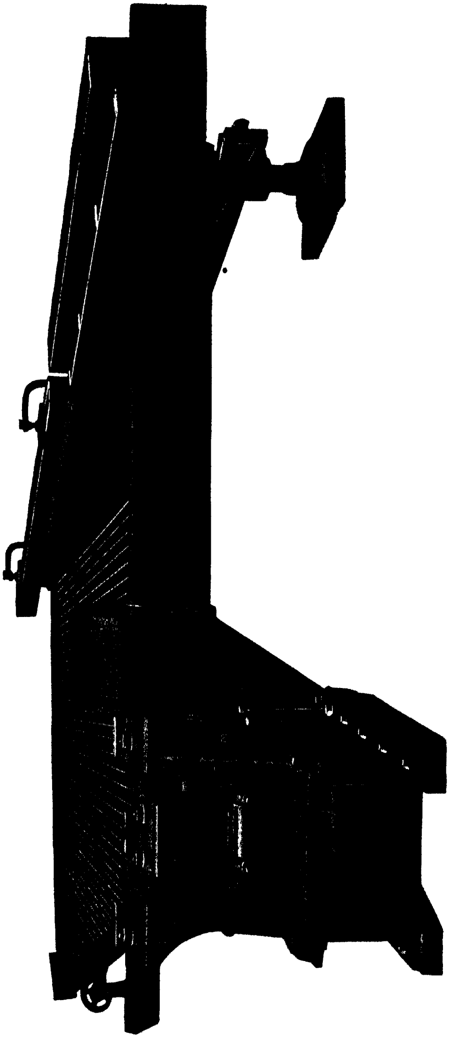

Fig. 287. Overstrom table. Perspective.

sizes, a larger for sands, and a smaller for slimes. The former is rectangular, 11 feet 6 inches by 5 feet, makes 340 strokes and requires from 5 to 9 gallons of clear water per minute; it takes ½ H.P. to drive it and its capacity is from 9½ to 14½ tons per 24 hours. Its weight is about 27 cwt. and its price is about £90.

The smaller table is trapezoidal, 9 feet long by 4 feet 6 inches wide tapering to 2 feet 3 inches; it makes 380 strokes and requires 2¾ to 3½ gallons of water per minute; it requires ⅓ H.P. to drive it, and it will treat 5 to 10 tons per 24 hours. Its weight is about 1 ton and its price about £67.

The **Overstrom Table,** Fig. 287, is carried on rollers, is rectangular, and has riffles sloping in the opposite direction to the Buss table, i.e. upwards from the motion end.

The **Bartlett Simplex Concentrator** consists, as shewn in Fig. 288, practically of three narrow Ferraris tables, placed one above the other, so that the material to be dressed undergoes treatment at three successive levels.

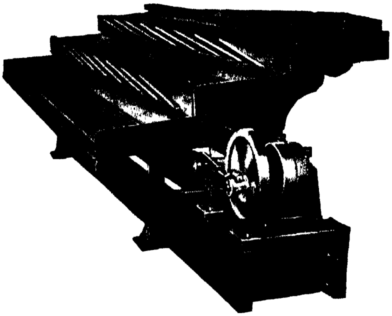

Fig. 288. Bartlett simplex concentrator. Perspective.

The surface of the table is covered with solid indiarubber in which the riffles are moulded, there being on each deck 13 riffles, 1 inch apart, ½ inch deep at the feed end running out to nothing at 9 inches from the discharge end.

The overall dimensions of the table are 12 feet 4 inches long by 5 feet 6 inches wide. It is usually run at 240 to 250 strokes per minute with a one inch or 280 to 300 half-inch strokes. It takes ½ H.P. to drive it, and will treat on an average 25 tons per day. It weighs about ½ ton and its price is about £70.

It is manufactured by the Colorado Iron Works Company of Denver, Colorado.

Another table of this class is the **Cammett Table** made by the Denver

Engineering Works and shewn in Fig. 289. The table is rectangular in plan, about 16 feet by 6 feet and is covered with painted canvas; it is supported on slides so as to swing steadily to and fro, the motion being given by a crank and slotted link, the table being kept up to its work by a spring. It is said to take $\frac{1}{4}$ H.P. to drive it, to be able to treat stuff from 3 mesh down to slimes, with a capacity of from 10 to 40 tons per 24 hours and a consumption of wash water ranging from 5 to 20 gallons per minute. It weighs, complete, about a ton, and costs about £90.

Several rotatory shaking tables have been devised, which may be said to bear the same relation to vanners or to the Rittinger table that the Linkenbach table does to the flat table or frame.

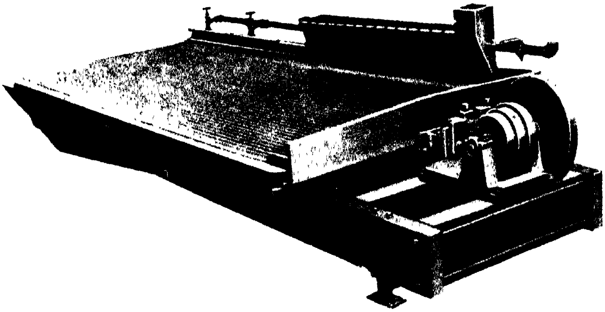

Fig. 289. Cammett table. Perspective.

The best known of these is the **Bartsch Table**[1], Fig. 290, which though it has not come into extensive use, is well spoken of. It is constructed like the Linkenbach table (see p. 330) with a rotating headboard, collecting gutters, water sprays, etc.; the table, however, instead of being fixed, is carried on rollers or springs and receives a series of tangential impulses or bumps, produced by the motion of a cam shaft, the cam thrusting the table steadily against a spring, which jerks the table back sharply, the direction of the jerk being opposite to that of rotation of the headboard and fittings. The headboard consists of a circular trough perforated for about $\frac{1}{3}$ of its periphery; the pulp escapes through the perforations and flows directly on to the table, down which it runs, the lighter portions running down almost radially; the heavier portions, flowing less rapidly, remain longer on the table,

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[1] *Zeitsch. f. Berg. Hütt. u. Sal.-Wesen*, 1893, p. 207; *B. u. H. Ztg.* Vol. LII. 1893, p. 175.

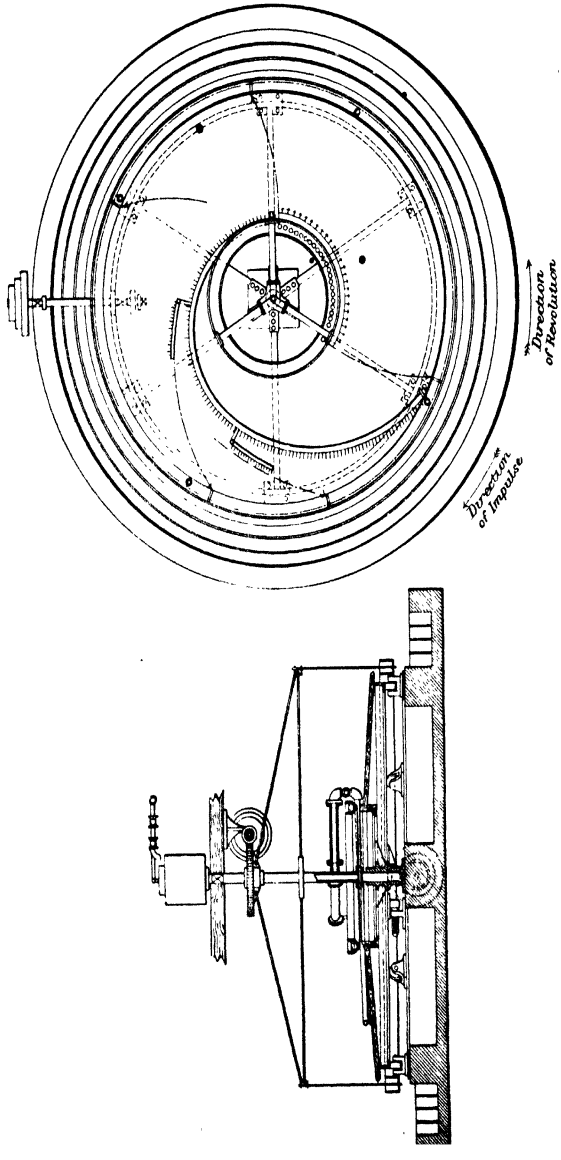

Fig. 290. Bartsch table.

and are therefore exposed for a longer period to the action of the impulses, which carry them to a greater angular distance from the point where they are fed on, before they flow off at the outer edge of the table. The action is thus practically that of a Lührig table in which the impulses are tangential, whilst the surface of the table rotates relatively to the position of the headboard, clear water feed and receiving launders, instead of being in linear motion. It is obviously unimportant as far as the principle of the machine is concerned, whether the surface of the table moves whilst the fittings are at rest, or *vice versa*. A Bartsch table 13 feet in diameter is capable of treating 9 to 10 cwt. per hour; it weighs about 5½ tons and costs about £200.

The **Sparre Table** was an earlier form, practically identical with the Bartsch table; it appears never to have come into extensive use.

A small machine, working on a somewhat similar principle, known as the **Hendy Concentrator,** was at one time extensively used in the Western States of North America; it has however gone entirely out of use.

CHAPTER X.

PNEUMATIC SEPARATION. MAGNETIC SEPARATION. ELECTROSTATIC SEPARATION. SEPARATION BY SURFACE TENSION.

PNEUMATIC SEPARATION.

PNEUMATIC separation is the term applied to the separation of bodies moving or falling in air, in contradistinction to hydraulic separation, where the medium in which the action takes place, is water.

It has been shewn in Chap. V, p. 220, that two spherical bodies will be equal-falling if their diameters and specific gravities are to each other in the ratio $\frac{D}{D_1} = \frac{S_1 - s}{S - s}$, where D, D_1 are the diameters, and S, S_1 the specific gravities of the respective particles, and s the density of the medium in which they fall. Therefore in an aggregate of particles of density S and S_1 (S being greater than S_1) and of all sizes ranging from the smaller diameter D to the greater D_1, all the particles of the heavier material will fall faster than any of those of the lighter, provided that D_1 is less than $D\frac{(S - s)}{S_1 - s}$, so that the smaller the fraction $\frac{S - s}{S_1 - s}$, the greater can be the difference in diameter of the particles without interfering with the completeness of the separation, or in other words, the more efficient the separation. When the medium through which the particles fall is water, this fraction becomes $\frac{S - 1}{S_1 - 1}$, whilst when the medium is air, it becomes $\frac{S}{S_1}$, and as the former fraction is necessarily smaller than the latter, it follows that pneumatic separation can never be as efficient as hydraulic separation, quite irrespective of the merits of the machines employed. Furthermore pneumatic separation is only possible when the particles are thoroughly dry, so as to have no tendency at all towards clogging or clinging together. Since there are very few mining districts that produce such absolutely dry material, the scope of the application of pneumatic separation is limited, unless the material to be treated be artificially dried, a process that is always a somewhat costly one. Finally the cost of dry crushing is always greater than that of wet crushing. Accordingly these three inherent drawbacks, the lower efficiency of the separation, the need for perfectly dry material, and the expense of dry crushing, have caused pneumatic concentration to be confined to certain practically rainless districts such as Western Australia, Arizona, etc., where the great scarcity of water makes it impossible to employ hydraulic separation.

The advocates of pneumatic separation claim that the greater mobility and lightness of air as compared with water is an advantage, as it enables lighter machines to be used, and these to be run at higher speeds; dry fine sands form a loose mass, readily penetrated by compressed air, whilst they pack tightly under the action of water, e.g. on a jig bed, so that much finer sands can be treated pneumatically than hydraulically; furthermore fine slimes are apt to be carried off by a stream of water that would be saved in air. They also point out that air forms a medium everywhere obtainable, not liable to freeze, and exerting no chemical action upon the minerals.

It is doubtful whether all these claims are well founded, and in any case it is certain that in practice they have been found not to outweigh the disadvantages above enumerated. The views held by most authorities on the subject of pneumatic concentration have been well summarised by Dr James Douglas[1] as follows: "the result has generally proved so much less perfect than that attained by wet concentration, and the maintenance of the machine in repair so much more costly, that the system, whatever support it may obtain from theory, has not made headway where water is available."

Pneumatic separation will therefore be but briefly considered here.

Various principles are employed in pneumatic separation, which are counterparts of those already considered under hydraulic separation. These principles are:

I. Submitting the material to be dressed to horizontal currents of air, when the lighter will be carried further before they fall to any given level.

II. Submitting the material to intermittent currents or puffs of air in a more or less vertical direction, when the heavier particles will fall against the puffs of air more rapidly than the lighter particles; this is usually spoken of as pneumatic jigging.

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[1] *Trans. Amer. Inst. Min. Eng.*, "American Improvements and Inventions in Ore-crushing and Concentration," by James Douglas, Vol. XXII. 1894, p. 328.

III. Projecting the material to be separated through the air, when the heavier particles will be carried a greater distance than the lighter ones—provided of course that the very fine dust, that has little or no tendency to fall, be removed.

IV. Keeping the mineral particles in a state of mobility by ascending air currents, and in this condition submitting them to bumping action on a shaking table.

A few examples of appliances employing each of these principles will be considered.

I. Methods depending on the application of horizontal air currents.

One of the most elementary methods is the so-called "Dry-blowing" formerly practised, e.g. in Western Australia[1], which consisted in tossing up the fine portions of auriferous alluvial into the air when a breeze was blowing; the wind carried away the lighter materials whilst the heavier gold dropped straight down and was caught. Some simple machines have been devised in which the steady action of a blower is substituted for the capricious effect of the wind; these take the form of a pair of bellows or a small handblower sending a blast across a tray furnished with transverse riffles, or some equivalent device, for collecting the heavier material, whilst the lighter is blown away.

Edison has used this principle in conjunction with magnetic separation for removing the lighter and finer particles, which contain a larger proportion of apatite, from the heavier grains of iron ore. He allows a stream of ore to fall in front of a series of practically horizontal blasts of air produced by a blower; the heavier grains fall almost straight down, being deflected but little from their paths by the blast, whilst the lighter is carried into dust chambers where it collects, or is blown straight out into the air.

A very similar arrangement was used in conjunction with an appliance known as the Niagara Pulveriser, a dry crusher, the product of which was separated into grades by an air-blast, carrying it through a number of different chambers.

The same principle has been applied to cleaning coal, especially in Germany.

Hochstraate's Apparatus at the Rheinpreussen Colliery for the treatment of fine coal is shewn in Figs. 291 and 291a[1]. It is applied to coal that has passed through a screen of 1·6 inch mesh; this is fed into a trommel *D*, making 6 sizes, the four finer ones being undersizes from 0·28 inch, 0·47 inch, 0·67 inch, and 0·87 inch respectively. Each of these

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[1] *Trans. Amer. Inst. Min. Eng.*, "The Alluvial Deposits of Western Australia," by T. A. Rickard, Vol. XXVIII. 1898, p. 503.

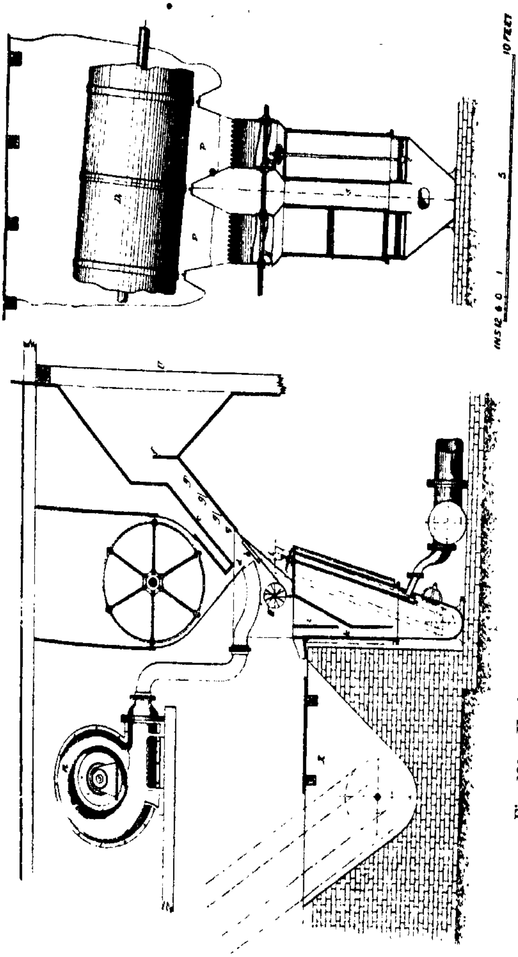

Fig. 291. Hochstraate dry coal cleaning machine. Side and end elevations.

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[1] *Zeitsch. f. Berg. Hütt. u. Sal.-Wes.* xxx. 1882, B. p. 280. German patents 3432 and 7959.

passes to a separate pneumatic apparatus. The entire mass of falling coal slides down an apron *P*, at the bottom of which it is caught by a blast of air coming from a fan *n* through a narrow slot *d*, and is carried thereby up a box *c* inclined at an angle of 60° and of such width that coal particles over 0·08 inch in diameter, together with correspondingly smaller particles of shale, etc., are deposited. Iron bars *g* are so placed that the blast is much feebler along the lower wall of the box than elsewhere, so that these particles can roll down, behind the partition *b*. At the end of the inclined box there is a vertical wall *f*, against which the flat scales of shale strike and drop down with the other particles. All these boxes open into a common air-chamber *e*, which is divided into three compartments, in which the fine coal dust is deposited, and whence it is removed by screw conveyors. The clean dust-coal thus got forms about $\frac{1}{5}$ of the whole, and is said to be specially valuable in the manufacture of coke. The coarser portions are treated in the hydraulic separators, the construction of which will be evident from the diagram.

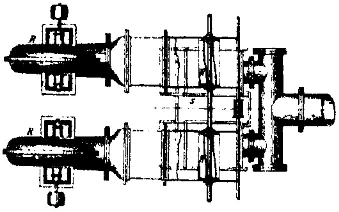

Fig. 291^a. Hochstraate dry coal cleaning machine. Plan.

An almost identical appliance is used at the Zollverein Colliery[1] in Essen, where dust up to 0·28 inch in size is blown out from the coarser coal by a Pelzer fan. It passes into a box inclined at 45°, down the bottom of which fine coal from 0·16 to 0·28 inch in size rolls, whilst all below 0·12 inch is carried into an air-chamber of 2,500 cubic feet capacity; the finest material settles here and is removed by a conveyor belt.

The Hochstraate apparatus has undergone some modifications, which are stated to have improved its efficiency. The improved form is shewn in Fig. 292[2]. The undersize from the trommel *F*, with screen of 0·28 inch mesh, drops at *A* past a blast issuing from an opening 80 inches by 4 inches; any coal that falls past this blast falls down the tube

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[1] *Zeitsch. f. Berg. Hütt. u. Sal.-Wesen.* Vol. xxxv. 1887, B. p. 264.

[2] *Ibid.* Vol. XLII. 1894, B. p. 235.

C and is exposed to a second blast at *A′* through an opening 160 inches long and 1·6 inches wide. The heavier particles continue their fall through *C* into the water-trough *D*; the lighter particles are as before carried by the blasts up the inclined boxes *B* and B_1 into the dust chamber *E*.

II. Very many machines working on the "air-jig" principle have been devised, but none seem to have come into permanent use.

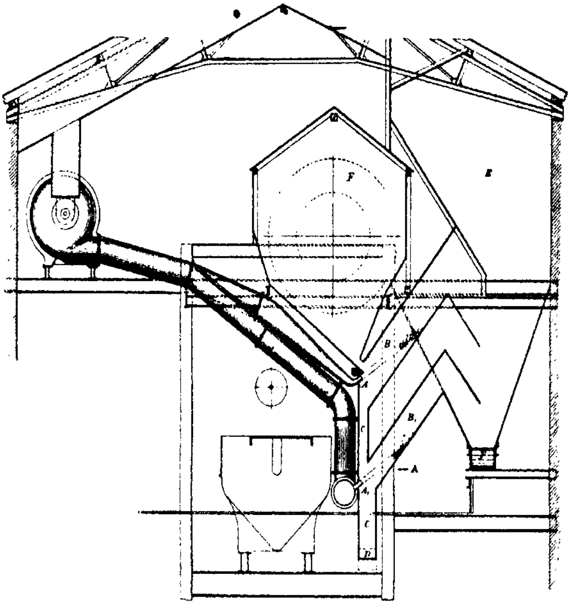

Fig. 292. Improved Hochstraate machine.

The **Paddock Air-jig**[1], Fig. 293, consists of an inclined bed formed of an iron grating over which a piece of stout cloth is firmly stretched, held down by a diagonal grating of brass strips, the distance apart of which varies with the nature of the material to be treated, and above this is a second grating, almost at right angles to the lower one. Below the bed are bellows worked by eccentrics, capable of ready adjustment, and giving from 400 to 500 puffs of air per minute. There is a hopper at the head of the bed through which the material to be treated is delivered on to the bed; under the action of the puffs of air, the heavier material settles into the lower grating by which it is guided into its discharge, whilst finger-bars separate out the middlings, and the lighter tailings are discharged separately, from the upper grating. The machine is said to have treated successfully material ranging from 35 up to 140 mesh; beyond the latter the machine did not give satisfactory results. The finer sizes can be treated at the rate of 1000 to 1200 lbs. and the coarser at the rate of 1500 to 2000 lbs. per hour.

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[1] *Eng. and Min. Journal*, Vol. XLII. 1886, p. 7; *Trans. Amer. Inst. Min. Eng.* Vol. VIII. 1879, p. 148 *Eng. and Min. Journal*, Vol. LIV. 1892, p. 130.

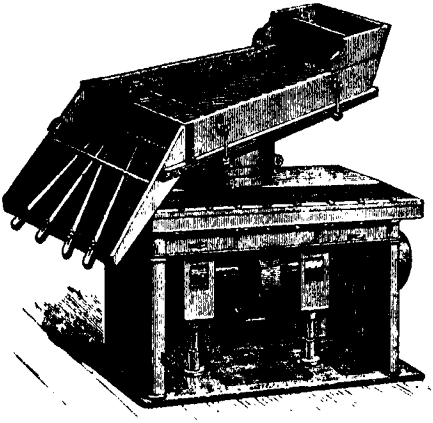

Fig. 293. Paddock air-jig. Perspective.

The **Krom Pneumatic Jig**[1], Fig. 294, consists of a bed composed of hollow bars of this section ⊓, made of brass wire sieving, placed from $\frac{3}{16}$ to $\frac{1}{4}$ inch apart, the lower portion of the interspaces being filled with strips of wood so that air can only enter through the hollow bars. The air is supplied in puffs at the rate of about 500 per minute from a small fan, which is suspended and caused to vibrate. The bed is about 6 inches in width; at one side of it the ore to be dressed is fed in through a hopper and in its passage across the bed the heavier portions settle down between the bars and fall into a reservoir below the bed, which is always kept full, material being discharged from it by a ribbed roller. The lighter tailings are discharged over a tailboard at the opposite side of the

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[1] United States Centennial Commission, International Exhibition, 1876. *Reports and Awards*, Group I. p. 302.

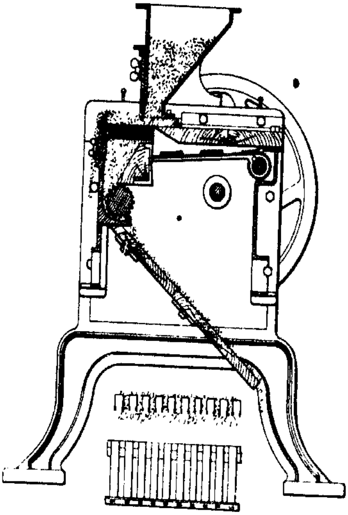

Fig. 294. Krom pneumatic jig.

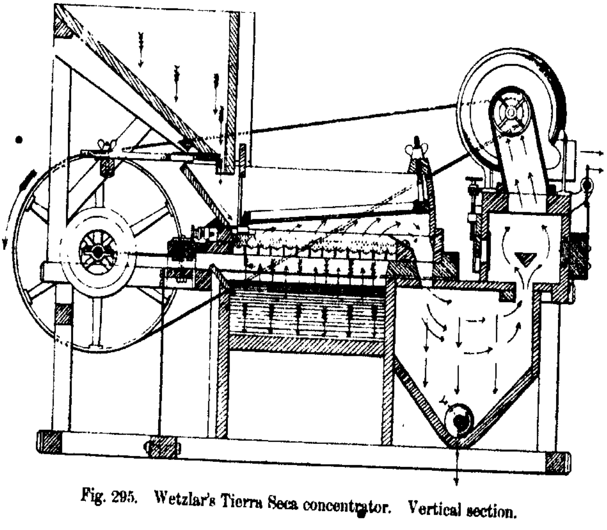

Fig. 295. Wetzlar's Tierra Seca concentrator. Vertical section.

screen. The machine weighs 1200 lbs. and is said to be capable of treating from 3½ to 12 tons per 24 hours and to require ⅛ H.P. It is said to be capable of treating material as fine as 140 mesh.

The **Wetzlar Tierra Seca** concentrator, Fig. 295, consists of a bed of corrugated perforated metal upon which rests a layer of suitable material, corresponding to the bed of an ordinary Harz jig; copper shot has been found to answer most purposes. An intermittent current of air is drawn through the bed by means of a fan. The material to be treated is delivered on to the bed from a hopper, and as it passes over the bed the heavier particles fall through the latter and accumulate in a box beneath, whilst the lighter tailings pass along and are discharged at the lower end of the bed. The largest size machine was stated by the inventor to have a capacity of 20 to 30 tons per 24 hours, and to require ⅔ H.P. to work it.

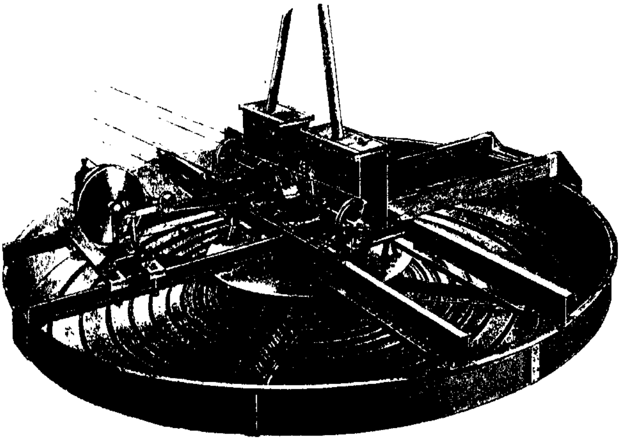

Fig. 296. Pape-Henneberg separator.

III. A few centrifugal separators have been tried, but appear not to have met with much success.

The **Pape-Henneberg** separator[1], shewn in perspective in Fig. 296, and in plan and vertical section in Fig. 296*, consists of a disc of steel plate, 18 inches in diameter, which revolves about a vertical axis at the rate of 2000 to 4000 revolutions per minute; over this is a fixed plate

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[1] *Oester. Zeitsch. f. Berg. u. Hütt.-Wesen.* 1893, p. 529; 1894, p. 68.

6 ft. 6 in. in diameter, through which the shaft of the revolving disc passes, the bearings being thus readily accessible. There are also openings in the upper disc through which the material to be treated is fed from hoppers on to the revolving disc. The latter is surrounded

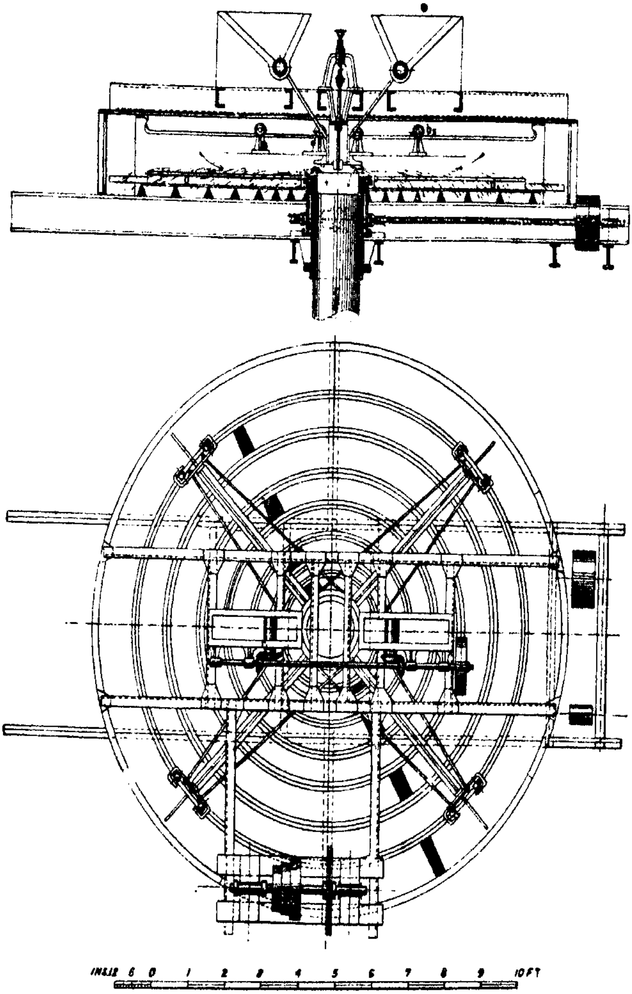

Fig. 296^{a}. Pape-Henneberg separator. Plan and vertical section.

by a number of concentric troughs, the largest being 20 feet in diameter. The ore that drops on to the rotating disc is flung outwards and is collected in the concentric troughs, the heaviest particles being in those of largest diameter. Rotating scrapers push the collected material through openings in the bottoms of the troughs. Beneath the rotating disc is a vertical pipe leading to an exhaust fan, which draws off all the very fine dust, and which, by creating an air current that opposes the motion of the particles, improves the separating action. The contents of each ring can be separated by sizing into heavier and lighter particles. The capacity of the machine is said to be 24 cwt. per hour with a power consumption of 3 H.P. This appliance has been tried in many places, notably in Western Australia, during the years 1891—95, but appears to have been altogether abandoned.

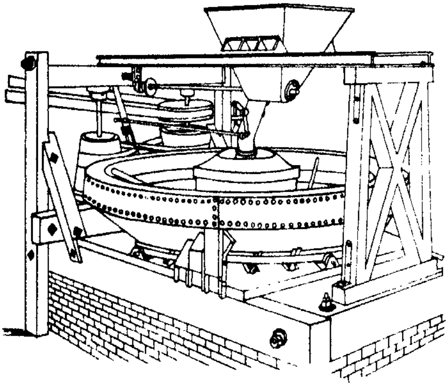

Fig. 297. Clarkson-Stanfield separator. Perspective.

The **Clarkson-Stanfield** concentrator, shewn in perspective in Fig. 297 and in diagrammatic section in Fig. 298, was tried for a while about the year 1890 in North Wales. As shewn in the figure, it consists of a grooved disc about 20 inches in diameter, revolving at a high speed about a vertical axis. Over it there is another disc connected with the lower end of a hopper from which the finely divided material, closely sized, drops on to the revolving disc. The particles are projected radially and those of highest specific gravity are thrown furthest from the machine before they drop; the different products are collected in annular troughs. It is stated that one of these machines 5 feet in diameter can treat 50 tons of mineral in 24 hours with a power consumption of 3 H.P.

IV. A certain number of appliances combine the action of a shaking table with that of an air-jig, or use puffs or currents of air to keep the mineral particles in a state of mobility, whilst the actual separation is effected mainly by the shaking action.

To this class belongs the **Sutton-Steele** table[1] shewn in longitudinal and transverse vertical section in Fig. 299. This is arranged somewhat like a Wilfley shaking table, the inclination from back to front being adjustable, whilst the throw is produced by a cam working against a bell-crank lever, and thus compressing a spiral spring that jerks the table back. The top of the table consists of wooden slats, beneath which is an air-chamber that receives a steady blast of air from a fan; upon these slats rests the true table top consisting of cloth readily pervious to air, whilst on this again are fastened tapering riffles, exactly like those of the Wilfley table. Different grades of cloth are used to correspond with the fineness of grain of the material treated, and the arrangement of the riffles can also be varied as desired. It is generally worked with a very low pressure of blast—about $\frac{1}{2}$ oz. to the square inch, and is run at about 400 impulses per minute. It is said to have given very good results in separating lead-zinc ores.

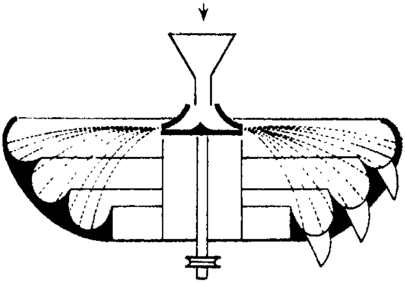

Fig 298. Clarkson-Stanfield concentrator. Diagrammatic vertical section.

MAGNETIC SEPARATION.

The possibility of the magnetic separation of minerals depends upon the manner in which minerals are affected when placed within a magnetic field. Faraday enunciated the important principle that bodies brought within a magnetic field will tend to move from places of weaker to places of stronger resultant force; further he shewed that, other things being equal, different bodies have different capacities for magnetic induction, so that the resultant force tending to produce motion depends in part upon the strength of the field—or more correctly upon the rate of variation of the strength of the field—and in part upon the magnetic properties of the bodies themselves.

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[1] *Eng. and Min. Journ.* LXXXI. 1906, p. 893. Brit. Pat. 17,561, 1905.

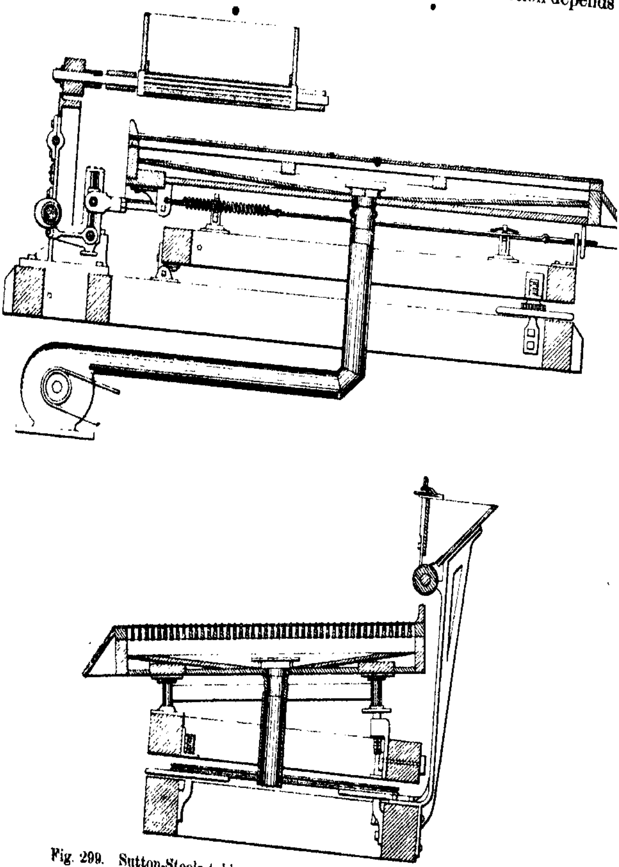

Fig. 299. Sutton-Steele table. Longitudinal and transverse sections.

The strength of a given magnetic pole is measured by the force with which it acts upon another pole at a given distance (the unit of strength being that of a pole thus capable of exerting a force of 1 dyne at a distance of 1 centimeter) or as it is often stated a magnetic field of unit strength is one that contains 1 line of force per square centimeter; the field strength is usually denoted by H. Such a magnetic pole is capable of inducing magnetism in a body placed within its magnetic field, and the magnetic flux or magnetic induction per square centimeter is usually denoted by B. The amount of this magnetic flux depends, other things being equal, upon a property of the body known as its permeability, in virtue of which it allows a greater or lesser number of lines of force to pass through it; the permeability of air is taken as the unit, and the permeability of other bodies is denoted by μ.

Hence $\frac{B}{H} = \mu$ and μ for air $= 1$.

The intensity of magnetisation of a body placed in a magnetic field or its magnetic moment per cubic centimeter (= pole strength per sq. centimeter) is denoted by I, and the force with which a magnet attracts the body may be taken as proportional to I, provided that it is permissible to neglect the effect which the body may exert upon the strength of the magnetic field; this is practically always the case in magnetic separation, where the strength of the magnetic fields employed is very great relatively to the magnetism induced in the particles of mineral. The magnetic susceptibility of a body is the ratio of the intensity of magnetisation to the strength of the magnetic field producing it, or denoting this susceptibility by κ, $\kappa = \frac{I}{H}$. The magnetic flux is due to both strength of field and intensity of magnetisation, so that

$$B = H + 4\pi I, \text{ or } \mu = 1 + 4\pi\kappa = 1 + 12{\cdot}566\kappa.$$

Under the ordinary circumstances, such as obtain for example in a well constructed electro-magnet, the permeability of good soft iron (the most permeable body known) is about 2000, but μ is not a constant, and for the same piece of iron under different conditions it may range from less than 2 to over 5000. In the case of iron, which has been most studied, I attains a maximum value with comparatively low values of H, so that μ increases rapidly at first as H increases, and then diminishes almost as rapidly as H continues to increase. The magnetic susceptibility of a body depends therefore not only on the substance itself but upon the conditions to which it is exposed; hence determinations of the susceptibilities of various minerals can only be taken as general guides to their behaviour in a magnetic field, the numerical determinations holding good only for the specific conditions of each experiment.

It can however be stated generally that if two particles of mineral of different magnetic susceptibilities be introduced into a magnetic field, they will tend to move towards the strongest part of that field with different degrees of force, the mineral of greater magnetic susceptibility being of course the more powerfully acted on. If this difference is sufficiently great, it may be used as a means of separating the minerals from each other; in practice it is usual to select a strength of field such that the force acting upon the more susceptible is sufficient to cause it to move, not infrequently in opposition to gravity, whilst that acting on the less susceptible is unable to overcome the resistance to motion offered by the particle.

It is therefore necessary first of all to know the relative magnetic susceptibilities of various minerals. Plücker in 1849 attempted to determine these for a few minerals and gives the following as the ratio of magnetic susceptibility of certain bodies as compared to iron:

Iron . .	100,000
Magnetite . .	40,000
Spathic iron ore	767
Haematite .	714
Specular iron ore	593
Limonite . .	296

An elaborate series of tests has been published by W. S. Crane[1] in which he has determined the tractive force exerted upon various powdered minerals by a powerful magnet. He shewed that under the conditions of his tests, the tractive force varied directly as the weights of mineral acted on, and that it varied directly with the magnetic flux or the intensity of magnetisation. He found that the mechanical condition of the material affected the permeability very greatly; thus wrought or cast iron in filings or in grains had a permeability of approximately $\frac{1}{3}$ of that of the same material in the form of bars, whilst with minerals of low susceptibility the tractive force is greater the smaller the size of the particles.

The majority of the tests were made with a field producing a magnetic flux of the order of 10,000, and for finely divided material in each case, crushed to pass 190 mesh; the tractive force exerted upon the substance, expressed in percentages of the weight of the substance, is also given. According to Mr Crane's experiments, $\kappa \propto \frac{\text{tractive force}}{\text{weight}}$, hence the susceptibility is proportional to these figures. The following are some of his more important results:

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. xxxi. 1902, p. 405.

Mineral	Permeability (μ)	Tractive force
(Iron	2·1617	72,605·81)
Magnetite	1·4669	29,140·00
Franklinite	1·4112	23,942·15
Ilmenite	1·2871	18,500·21
Pyrrhotite	1·0782 – 1·0775	4898·48 – 1358·80
Zircon	1·0293—1·0019	513·90—33·61
Haematite	1·0242 – 1·0081	423·04—149·90
Corundum	1·0253 – 1·0018	443·42—32·87
Siderite	1·0234 1·0218	452·98—378·05
Rhodonite	1·0176	840·45
Limonite	1·0099	174·10
Pyrolusite	1·0088 – 1·0078	155·34 – 136·85
Pyrites	1·0064 1·0007	112·91—12·33
Zinc blende	1·0057 – 1·0007	101·00 13·51

For all other minerals μ is on the average under 1·003, and for the purer specimens probably considerably lower. In most cases where there are marked variations in the permeability of specimens from different localities, this is probably due to variations in the percentage of iron present, either as a constituent of the mineral or as an impurity; this is very probably the case with zircon and corundum.

Very great variations are however observed in some cases where this explanation is not sufficient; for example some specimens of ilmenite (or at any rate of titaniferous iron ore) are so feebly magnetic, that magnetite may be separated from them so completely that the resulting concentrate contains hardly a trace of titanic acid; whilst with other specimens (e.g. from Taberg in Småland, Sweden) it is impossible to effect any magnetic separation between these minerals, and the concentrate is richer in titanic acid than the crude ore.

Such variation is shewn also in the determination of the magnetic permeabilities and susceptibilities of a few minerals from different Swedish localities by E. Holm[1], the highest and lowest values being as follows:

Mineral	Permeability (μ)	Susceptibility (κ)
Magnetite	3·62—5·71	0·208—0·375
Specular iron ore ...	1·0122– 1·0192	0·000974—0·00152
Garnet	1·00404—1·00844	0·000322—0·000672
Pyroxene	1·00164 – 1·0328	0·000130—0·00261
Hornblende	1·00231—1·0112	0·000164—0·000891

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[1] *Jernkontorets Annaler*, "Undersökning öfver de magnetiska egenskaperna hos några i svenska järnmalmer ingående mineral," 1903, p. 363.

These tests were made in a field the strength of which was of the order of 1200 to 1800 C.G.S. units. Further, Mr Holm has shewn that for many minerals also κ is variable for different values of H.

Thus he gets the following data for a sample of magnetite:

3·02	12·7	1·01
13·4	12·6	1·00
41·9	15·7	1·25
458	10·4	0·749
1170	5·71	0·375

indicating that magnetite behaves in a somewhat similar manner to iron in this respect. Nothing conclusive is known as to the behaviour of bodies of low magnetic susceptibility. It appears however to be tolerably clear that extremely powerful fields are prejudicial to the separation of bodies of high magnetic susceptibility. It need hardly be said that in modern practice the magnets used for magnetic separation are always electro-magnets, the strength of field of which depends upon the strength of the current and the number of the wire windings through which it passes, as long as the limit of saturation of the iron cores of the magnets is not exceeded.

The author has found in one experiment that the crystallised phosphate of iron, vivianite, had a magnetic susceptibility of about 0·6 compared to specular ore taken as 1, and of about 0·005 compared to magnetite.

It will be noted that there are but few minerals that are strongly magnetic in their natural state. It is however possible to convert several others (all containing considerable percentages of iron), which in their natural state are but feebly magnetic, into strongly magnetic forms by suitably heating them. This operation may either be simple heating in a neutral atmosphere or heating in a reducing atmosphere. As examples of the former, spathic iron ore, iron pyrites and chalcopyrite may be mentioned. When spathic iron ore ($FeCO_3$) is heated, it is decomposed, carbonic acid is given off, and a strongly magnetic residue is left, having according to Dr Wedding the composition Fe_6O_7.[1] Similarly iron pyrites (FeS_2), which is very feebly magnetic, is transformed by heat into a lower sulphide having the composition $Fe_{11}S_{12}$, which is strongly magnetic. According to some authorities an oxysulphide is produced, which is magnetic, but the subject has not yet been fully investigated. A very similar change takes place with copper pyrites, and it even appears possible that in both these minerals the change produced by heating may be physical quite as much as chemical. These changes can be produced by heating to temperatures not exceeding 300° C.

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[1] *Bull. de la Soc. de l'Ind. Min.* Ser. 3, Vol. IV. 1900, p. 1201.

Dr Wedding[1] states that on strongly heating haematite it loses some of its oxygen and is converted into magnetic oxide, which is of course strongly magnetic. The latter change is however in practice generally produced by heating in a reducing atmosphere; when ferric oxide (either in the form of red or of brown haematite) is heated in a reducing atmosphere, or in contact with carbonaceous matter, it is reduced to magnetic oxide at a low temperature, and the mineral is thus rendered strongly magnetic. This process has been successfully carried out on fossiliferous ore (red haematite) at Birmingham Ala[2], where the ore, broken to about egg size, was heated in a gas-fired Davis-Colby kiln, using ordinary producer gas; the kiln would treat 110 tons per day of 24 hours, consuming 390,000 cubic feet of gas produced from 3 tons of coal, and the results are stated to have been highly satisfactory. The magnetic separation of these artificially magnetised minerals differs in no wise from the treatment of naturally magnetic minerals.

For practical purposes it is convenient to divide minerals into three groups, according to their magnetic permeability, namely:

I. Strongly magnetic, including magnetite, franklinite, ilmenite and pyrrhotite, capable of being attracted and lifted by an ordinary permanent magnet (also including the above-mentioned mineral substances when artificially produced).

II. Feebly magnetic, including the more magnetic specimens in Mr Crane's list, p. 389, say with $\kappa > 0{\cdot}0002$, incapable of being attracted and lifted by an ordinary permanent magnet, but capable of being attracted by a suitably arranged very powerful electro-magnet.

III. Non-magnetic, or practically incapable of being attracted and lifted by any ordinary electro-magnet.

It is clear that the minerals of Group I can be separated magnetically from those of Groups II and III without any difficulty, and that minerals of Group II can by special arrangements be separated from those of Group III. There are accordingly two sets of processes, namely processes adapted to the separation of strongly magnetic minerals, and processes, adapted to the separation of feebly magnetic minerals. For the former either wet or dry methods may be used, for the latter none but dry methods have hitherto been found practicable.

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[1] *loc. cit.*

[2] *Trans. Amer. Inst. Min. Eng.*, "Notes on the Magnetization and Concentration of Iron Ore," by W. B. Phillips, Vol. xxv. 1896, p. 399.

Generally speaking dry methods can only be applied to minerals that have been artificially dried. It must not be forgotten that this fact sometimes gives rise to unexpected difficulties; thus it is easy to separate magnetite from iron pyrites by the wet magnetic method so as to produce a concentrate practically free from sulphur. On drying such a mixture, however, it is practically almost impossible to so regulate the temperature that none of the iron pyrites shall be decomposed; a certain amount of the sulphur is always driven off, producing magnetic pyrites, which passes into the concentrate and contaminates it with sulphur.

It must be noted that the effect of a suitable magnetic field upon a mixture of magnetic and non-magnetic particles is merely to attract the former, so that the sustaining capacity of the magnetic field would soon be reached unless some provision were made for removing these particles and allowing others to take their place; this may be done (1) by causing the magnetic fields themselves to travel; (2) by interposing between the magnets and the particles a travelling non-magnetic surface, which carries the particles for a certain distance through the magnetic field sensibly parallel to the lines of force, under which conditions the particles will adhere to the non-magnetic surface until the motion of the latter has carried them into a weaker part of the field, where they will drop off under the action of gravity or may be thrown off by centrifugal action; the surfaces in question may be either drums or belts; (3) the particles may be projected or be allowed to fall, and caused to pass through a magnetic field in such a manner that the magnetic ones are drawn aside from the normal path sufficiently to separate them from the non-magnetic ones; machines on this principle are often spoken of as deflection machines. In dry magnetic separators the non-magnetic particles are either carried away by belts or drums or are allowed to drop away from the magnetic field under the action of gravity; in wet magnetic separators this removal is often assisted by a current of water. These latter machines are therefore generally the more efficient, inasmuch as the stream of water is more effective in washing away non-magnetic particles that may have become entangled among the magnetic ones. Some dry magnetic separators use a blast of air for the same purpose, and thus act to some extent like pneumatic concentrators.

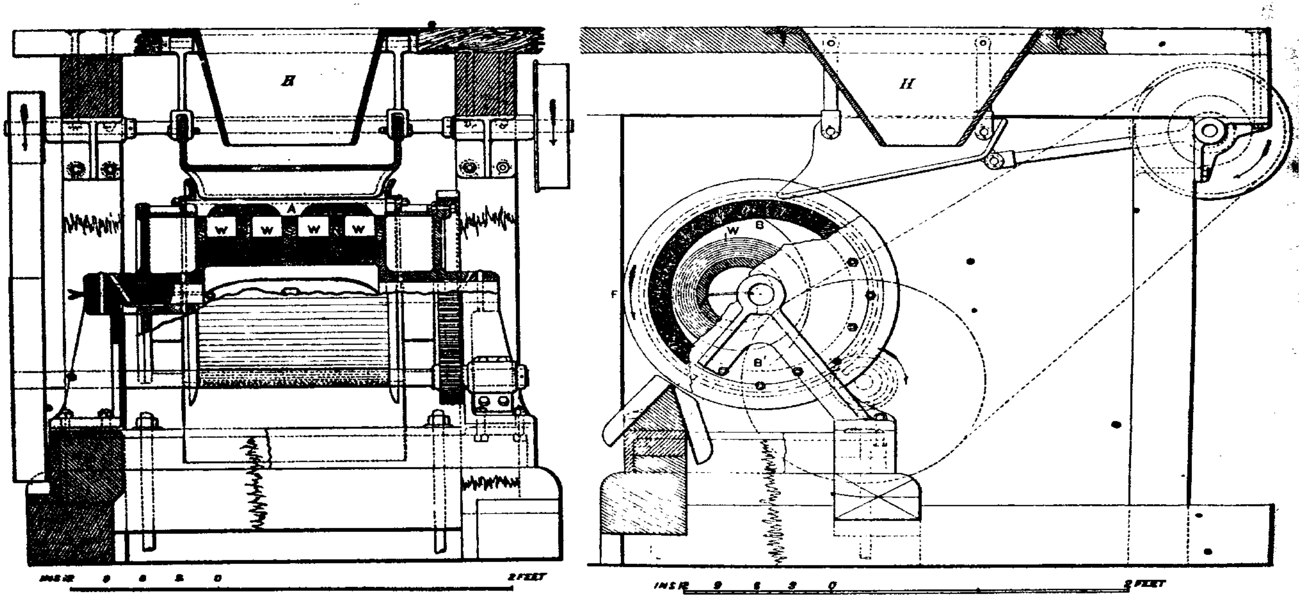

Fig. 300. Wenström separator. Sectional longitudinal and end elevations.

I. CONCENTRATION OF STRONGLY MAGNETIC MINERALS.

A. *Dry methods.*

1. Machines employing moving magnets.

One of the earliest and most efficient of these machines, which is still largely used, is the **Wenström** machine[1] devised in Sweden in 1883. It is shewn in sectional elevation in Fig. 300 and in perspective in Fig. 301.

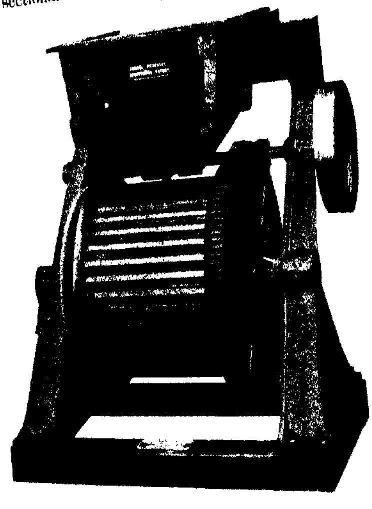

Fig. 301. Wenström separator. Perspective.

It consists of a horizontal drum built up of alternate bars of wood and soft iron; the latter have projections at the back, alternate bars having respectively two and three such projections (*A*, Fig. 300). A stationary electro-magnet in the form of a hollow cylinder, *E*, is placed horizontally nearer to the front than to the back of the drum. This electro-magnet is furnished with five pole pieces, of the shape shewn in Fig. 300, the front part *B*, *B'* being curved to fit the inside of the drum. The electro-magnet is so wound that these pole pieces are of opposite polarity as indicated by the letters *N* and *S* in Fig. 300; the projections at the back of the soft iron ribs of the drum are thus in contact alternately with one or the other set of pole pieces and are therefore alternately of *N* or *S* polarity as long as they are travelling (in the direction of the arrow) from *B* to *B'*. The ore to be treated is tipped into the hopper *H* and is fed on to the drum of the machine by a simple shaking tray; the non-magnetic material rolls over the face of the revolving drum and drops into the front shoot; the magnetic material adheres to the barrel and is carried round with it until it has passed the point *B'*, where the iron bars pass out of contact with the pole pieces of the electro-magnet, and are thus no longer themselves magnets, and accordingly release the magnetic particles, which drop into the rear shoot. The machine is made in two sizes; the smaller, having a drum about 20 inches in diameter by 15 inch face and taking a current of about 10 amperes at 35 volts, will treat from 2 to 3 tons per hour in pieces not exceeding ¾ lb. in weight; the larger, with a drum about 30 inches in diameter by 24 inches face, will treat up to 7 tons per hour, in pieces not exceeding 7 lbs. in weight; on finely crushed ore its capacity is rather less, say about 5 tons per hour. The drum makes 30 revolutions per minute, and the current required is 15 amperes at 110 volts. In addition to the power needed to generate the electric current, about ½ H.P. is required to drive the drum.

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. XVII. 1888, p. 599; *Eng. and Min. Journ.* 1888, Vol. XLVI. p. 437.

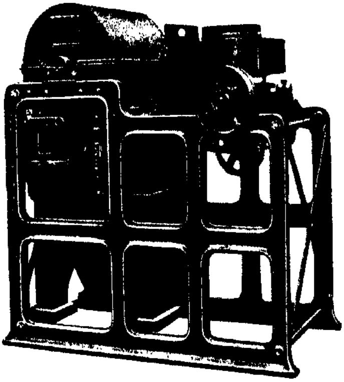

Fig. 302. Humboldt coarse magnetic separator. Perspective.

This machine is one of the very few magnetic separators capable of treating lump ore, for which purpose it is still extensively used.

A machine on quite the same principle has been brought out recently by the **Humboldt** Engineering Works Co., the appearance of which is shewn in Fig. 302. The material is fed by the shaking tray shewn at the right hand side into a casing in which it drops upon a rotating drum made of brass or similar non-magnetic material; from this drum it falls in a uniform stream and with a velocity equal to that of the magnetic drum, which is constructed very like that of the Wenström machine; the latter has no effect on the non-magnetic particles, which drop straight down off the brass drum; it attracts the magnetic particles and carries them round with it till they pass out of the magnetic field when they drop off, less magnetic middlings not being carried as far as the more magnetic portions, so that this machine yields three products, namely, highly magnetic concentrates, moderately magnetic middlings and non-magnetic tailings. These machines have been used for concentrating calcined spathic iron ore in the Siegen district. The magnetic drum is about 2 feet diam. by 2 feet long, and the electro-magnet inside it is wound with 5000 turns, through which a current of 5 amperes flows, a tension of 70 volts being sufficient for this purpose. It appears to be capable of dealing with about 20 tons per 10 hour shift.

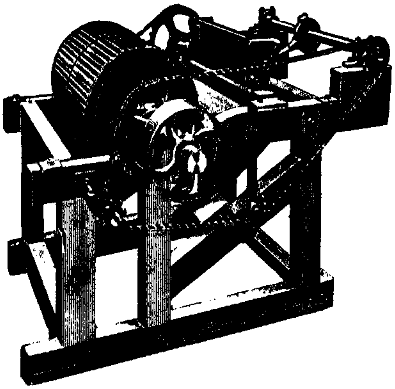

Fig. 303. King magnetic separator. Perspective.

A somewhat primitive machine, **King's Patent Magnetic Ore Separator,** Fig. 303, consists of a somewhat similar drum revolving over a stream of mineral fed towards it on a shaking tray. In this drum, however, permanent magnets are employed instead of electro-magnets. These machines have been used in Namaqualand for removing magnetite from copper ores. A machine with a drum of 26 inches face and 20 inches in diameter is said to treat up to two tons per hour of moderately fine stuff and to require about 1 H.P. to drive it; these permanent magnets cannot be made powerful enough to pick up heavier pieces of mineral.

The **Gröndal Coarse Separator** is shewn in plan in Fig. 304 and sectional elevation in Fig. 304ᵃ. It will be seen that it consists of a conveyor belt which goes round and is driven by a drum, which effects the actual separation; the machine may therefore be described as a combination of a belt conveyor and a drum separator. It will be seen that inside the drum, which is made of brass or some similar non-magnetic material, there are arranged four fixed electro-magnets which produce a strong magnetic field through an angle of about 60° above and 60° below the horizontal plane that passes through the axis of the drum, this being the driving portion of the drum against which the belt rests. The

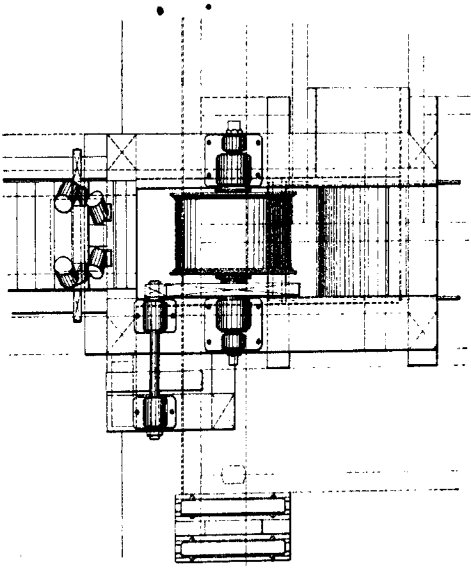

Fig. 304. Gröndal coarse magnetic separator. Plan.

broken mineral is carried forward by the belt, and whilst the non-magnetic material drops straight off the face of the drum, the magnetic material adheres to it and is carried round further with it, dropping off at the bottom of the drum. The division betwfeen the two kinds is capable of adjustment by moving the hinged partition shewn in the section. This machine can treat 6 to 12 tons per hour of ore broken to a 2 inch ring; the current required is 7½ amperes at 110 volts. With a belt 100 feet long set at a grade of 18°, the driving power required is 5 to 6 H.P. The machine costs about £150.

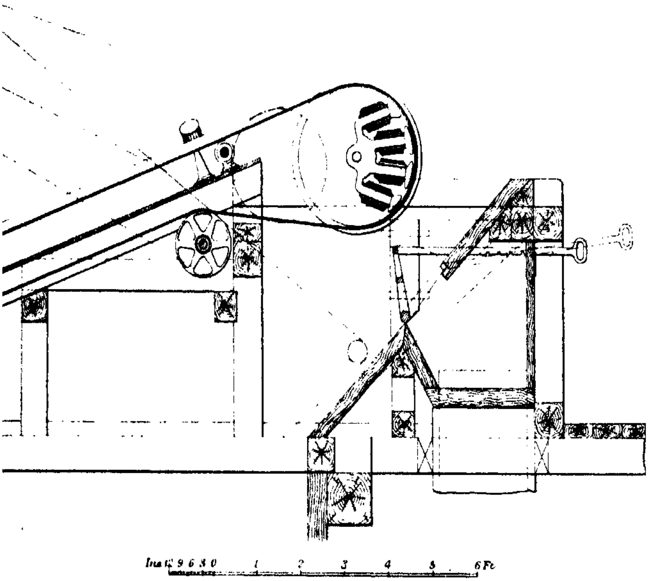

Fig. 304ª. Gröndal coarse magnetic separator. Vertical section.

Kessler's[1] **Magnetic Separator,** Fig. 305, has been used for separating calcined spathic ore from blende, crushed from 0·04 to 0·2 inch. This is fed by a feed roller on to a belt furnished with pins of soft iron, which passes over a wooden roller at the head end and a cylindrical electro-magnet at the lower end. The non-magnetic material drops off at the end of the belt, whilst the magnetic portion adheres to the pins, which retain their magnetism as long as they are within the field of the cylindrical electro-magnet, and is thus carried to some little distance back from the end of the belt, where it is discharged beyond an adjustable partition.

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[1] *Zeitschr. f. Berg. Hütt. u. Sal.-Wesen.* XLII. 1894, B. p. 232.

One of the earliest machines used in practice was devised in 1858 by **M. Sella**[1] at the well-known Traversella mines in Northern Italy, in order to separate magnetite from copper pyrites. The finely divided and well-dried ore was fed on to a travelling belt 15 inches wide carried on 3 inch rollers; over this revolved three wooden discs 20 inches in diameter keyed to a horizontal shaft parallel to the travel of the belt, making about 10 revolutions per minute. Each wooden disc carried 18 electro-magnets 6 inches long, 5 inches wide, 1 inch thick, set so as to alternate. These magnets picked up the magnetite and after the drum had revolved through about 75°, the electric current (here produced by six large Bunsen cells) was interrupted by means of a special commutator, when the magnetite dropped off into a suitably disposed shoot. The machine was capable of treating per hour about 3 cwt. of ore, which was separated into about ⅔ magnetite and ⅓ copper ore.

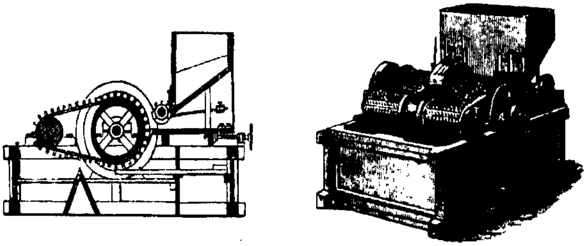

Fig. 305. Kessler's magnetic separator. Vertical section and perspective.

Ding's[2] **Magnetic Separator,** Fig. 306, acts somewhat like the last-named machine; the ore is fed from a hopper *A* on to a shaking tray *N* 16 inches wide, down which it slowly passes; above the tray revolve wheels of aluminium with soft iron studs on their peripheries, the diameter of the wheels being somewhat greater than the width of the tray. An electro-magnet is so arranged as to magnetise the studs during the lower half of their travel, by which means the magnetic particles in the ore are lifted up and dropped clear of the tray into the bins *E*; the non-magnetic portion is discharged at *G*. The electric power required is stated to be 1 ampere at 110 volts and the working capacity about 1 ton per hour.

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[1] *Zeitschr. f. Berg. Hütt. u. Sal.-Wesen.* Vol. IX. 1861, p. 172; *U.S. Centennial Commission Report*, 1876, p. 313.

[2] *The Mineral Industry*, Vol. XII. 1903, p 417; Vol. XV. 1906, p. 830.

One form of Ferraris magnetic separator acts on a somewhat similar principle.

Putzig's Machine[1] consists of a disc revolving on a vertical shaft; to the lower face of the disc are attached some 20 electro-magnets, the lower poles of which pass through a trough in which the ore to be treated is contained, this trough forming about $\frac{2}{3}$ of a circle; when the magnets pass beyond the trough, the current is cut off and the magnetic portion, which had been attracted by them, drops off into a suitably placed receiver. The machine has been improved in many respects by Messrs Siemens and Halske.

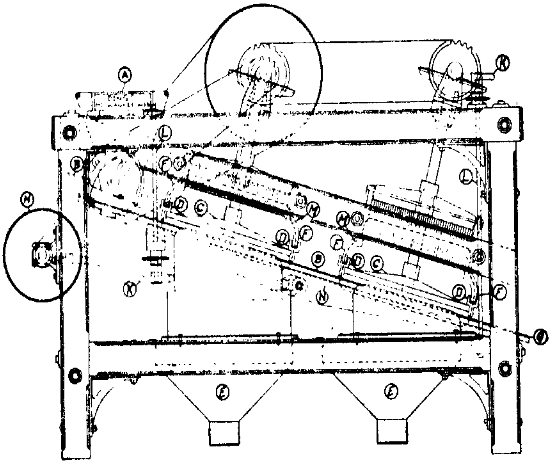

Fig. 306. Ding's magnetic separator. Longitudinal elevation.

2. Machines with fixed magnetic poles are very largely used, the magnetic mineral being carried through the field either by drums or belts.

The **Buchanan**[2] **Magnetic Separator**, an older pattern of which is shewn diagrammatically in Fig. 307, and in its more modern form in Fig. 308, consists of a pair of soft iron rolls, each supported upon the similar pole pieces of a couple of horseshoe magnets, so that the rolls

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[1] N. Wedding, *Bull. Soc. Ind. Min.* Vol. XIV. 1900, p. 1217.

[2] *Trans. Amer. Inst. M. E.* Vol. XVII. 1888–9, p. 737.

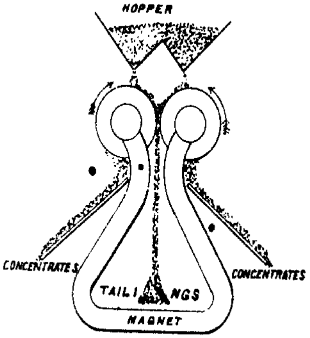

Fig. 307. Buchanan magnetic separator. Diagram.

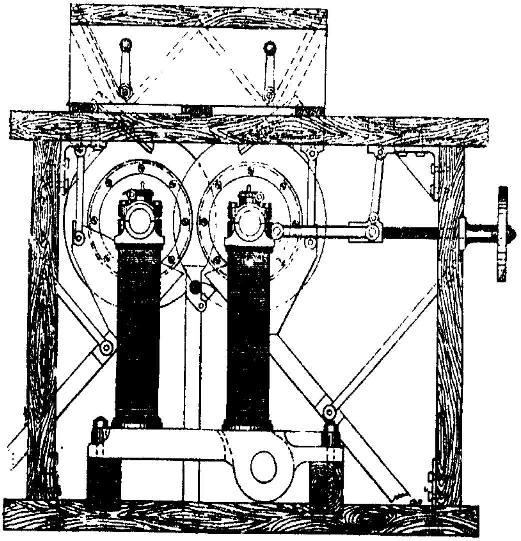

Fig. 308. Buchanan magnetic separator. Elevation.

receive opposite magnetic polarity. There is thus a tolerably strong field formed between the two rolls. The material to be treated is allowed to fall between the rolls which revolve towards each other at a speed of about 150 feet per minute; the non-magnetic portion continues to fall without interruption, whilst the magnetic particles adhering to the rolls are carried by their revolution out of the stronger portion of the field, until they drop off or are thrown centrifugally off the rolls. In the modern form two of the magnets swing on trunnions and thus enable the space between the rolls to be adjusted.

One of the more extensively used of these drum machines is the **Ball-Norton Monarch Machine,** a modern form of which is shewn in Fig. 309[1]. It consists of two drums, *A* and *B*, of wood or some similar non-magnetic material, rotating on horizontal shafts in the directions indicated. The electro-magnets, as shewn, occupy from one to two thirds of the lower portion of the circumference of the drums; they are arranged radially, and in such a way that opposite poles alternate. The drums are placed in a closed wooden chest, that can be connected at *S* with a powerful fan and having an opening at *E* through which a current of air can be drawn in by the action of the fan. The ore to be treated is fed in through the hopper *H* and falls upon drum *A*, the rotation of which brings the stream of ore within the magnetic field of the first set of magnets; the non-magnetic portion drops into the hopper *P* whence it is discharged intermittently by the weighted trap-door *F*. The magnetic portion is carried round with the drum until it is thrown off at the edge of the magnetic field, when it falls upon an apron, and is thus brought within the magnetic field of the second drum. By either

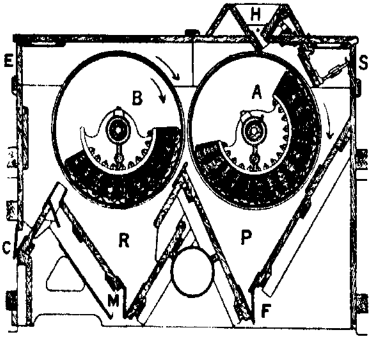

Fig. 309. Monarch magnetic separator. Vertical section.

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[1] C. M. Ball, *Trans. Amer. Inst. Min. Eng.* Vol. xix. 1890-1, p. 187; *Jernkontorets Annaler*, "Om Anrikning af svenska Jernmalmer," by W. Petersson, 1903, p. 255.

making this field weaker or the speed of rotation of the drum greater, middlings are here thrown off and accumulate in the hopper *R* to be discharged through *M*. Finally the concentrates are discharged at *C*. The inventor claims that the arrangement of his pole pieces causes the magnetic particles to roll over repeatedly whilst being carried round by the drums, and that this rolling motion combined with the action of the air blast effectually frees them from non-magnetic particles. It is stated that a machine with drums 24 inches in diameter and 24 inches face will treat per hour 15 to 20 tons of ore crushed to 16 to 20 mesh; the power required is stated as 1 to 1½ electric H.P. for each drum and ½ to ¾ H.P. to drive the machine or say about 3½ H.P. altogether. This machine has been tried at several places in Sweden, but has been set aside usually for wet separators. At Svartön[1], in Northern Sweden, two Monarch machines

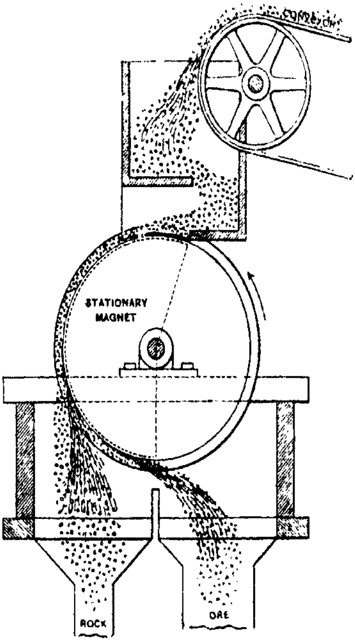

Fig. 310. Ball magnetic separator. Vertical section.

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[1] Walf. Petersson, *Jernkontorets Annaler*, 1903, p. 290.

treated together about 300 tons per day of 24 hours. The drums were of brass 20 inches in diameter and 24 inches long; the first drum made 30 revolutions per minute and the second 90; each drum received an electric current of 7 amperes and 100 volts.

The concentrator, sometimes called the **Sautler**, employed at Friedrichsegen[1] in Oberlahnstein is practically the same as a single drum Monarch machine, with the magnets placed along the side instead of the bottom of the drum and the ore fed on by a shaking table; it is used to separate calcined spathic ore from zinc blende and its working capacity is stated to be 2 tons per hour and its current consumption about ¼ H.P. A modified form of this machine, suitable for coarse material, has already been described (see p. 396).

Another practically identical machine is the **Ball Magnetic Separator**, Fig. 310, the ore being brought to the drum in this case by a conveyor belt: it is intended to treat moderately coarse ore.

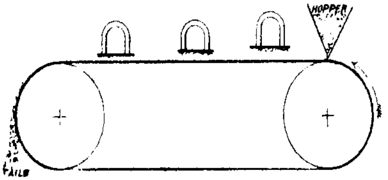

Fig. 311. Conkling magnetic separator. Diagram.

The **Siemens and Halske** and the **Heberle** dry separator are also somewhat similar.

One of the earlier forms of belt machines is the **Conkling** machine[2], shewn diagrammatically in Fig. 311. It consists of a carrying belt upon which the ore is fed, which conveys it under a series of electro-magnets, of which the strength, as well as the distance above the belts, can be adjusted as required. Under each electro-magnet runs a belt at right angles to the main belt; the magnetic particles as they pass into the magnetic fields are attracted upwards to these cross belts and are carried by them sideways clear of the main carrying belt, when they are deposited. By making the magnets of varying strength, both concentrates and middlings can be produced.

One form of Ferraris separator is somewhat like this machine.

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[1] *Jernkontorets Annaler*, 1902, p. 13. D. R. P. 24976, 1883; *Berg. u. H. Ztg.* 1884, pp. 167, 412.

[2] *Trans. Amer. Inst. Min. Eng.* Vol. XVII. 1888—9, p. 739.

The **Ball-Norton Belt Machine** is shewn diagrammatically in Fig. 312. The crushed ore is fed from a hopper by means of a feeding roller on to a travelling belt about 9 inches wide; above it and parallel to it is a similar belt travelling parallel to the lower one, so that the lower face of the former, and the upper face of the latter move in the same direction; above this upper belt are a number of electro-magnets, so wound that opposite poles alternate successively, as in the drum machine of the same name. The non-magnetic material remains on the lower

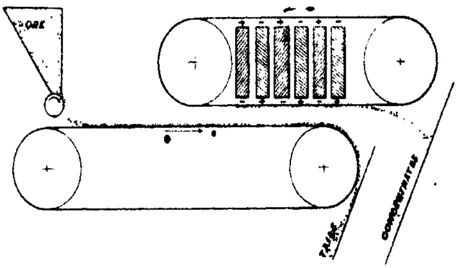

Fig 312. Ball-Norton belt separator. Diagram.

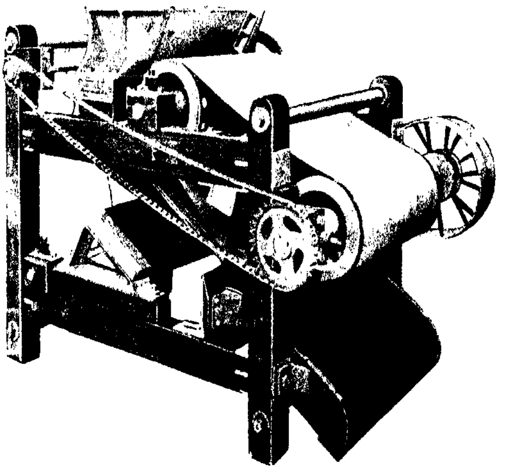

Fig. 313. King belt separator. Perspective.

belt, the magnetic material is attracted to the underside of the upper belt and carried by it in the magnetic field until it is clear of the lower belt, when it drops off. The machine can treat about 20 tons per hour.

The **King Electro-Magnetic Belt Machine** is not unlike the last in principle except that only the upper belt is represented, the lower belt being replaced by a shaking tray that feeds the ore forward. It is said to be capable of treating 1 ton per hour with a current consumption of 6 amperes at 100 volts, whilst 1 H.P. is required to actuate the machine.

3. The only true deflection machine is the **Edison Magnetic Concentrator**, the principle of which is well indicated by the accompanying diagram, Fig. 314, taken from Edison's paper[1]. As will be seen the ore is allowed to fall down past the face of a powerful magnet; the magnetic particles on entering the field of the magnet are drawn aside from the vertical line of fall, and thus are made to drop into one receiver, whilst the non-magnetic particles fall straight down into another. Such a simple experimental machine, with a magnet 72 inches by 30 inches by 10 inches, weighing 3400 lbs., wound with 450 lbs. of copper wire and taking a current of 10 amperes at 116·5 volts can treat per day about 150 tons crushed to

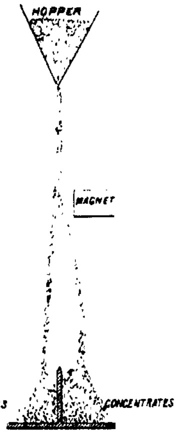

Fig. 314. Single Edison magnetic separator. Diagram.

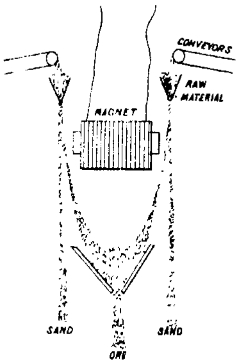

Fig. 315. Double Edison magnetic separator. Diagram.

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[1] *Trans. Amer. Inst. M. E.*, "The Concentration of Iron Ore," by J. Birkinbine and T. A. Edison, Vol. XVII. 1888-9, p. 728.

10 mesh. Edison has pointed out that the opposite pole can be utilised in exactly the same way, as shewn in his subjoined sketch, Fig. 315[1]. This process was first applied in New Jersey and subsequently at Dunderland in Norway. The following Figure shews diagrammatically the most recent construction of the bank of magnets used for extracting magnetite at the latter place; one such bank is used for extracting the magnetite and the other for cleaning the crude concentrate. The electro-magnets

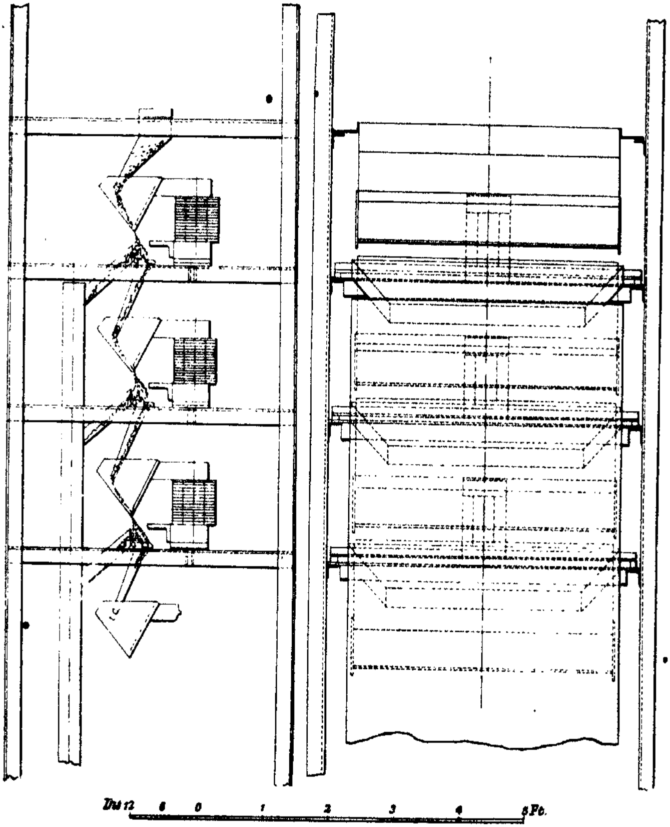

Fig. 316. Bank of Edison magnets. Side and front elevations.

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[1] Pamphlet issued by the Edison Ore Milling Syndicate, Ltd.

are arranged as shewn upon an iron framework, and the poles are widened out horizontally to a width of 4 feet. The ore is fed in from a hopper at the top of the bank of magnets and falls over baffle boards in a thin stream past the poles of the first magnet. Between these pole pieces a powerful magnetic field is produced which deflects a portion of the magnetic material, an adjustable dividing board being so placed as to separate the two streams of mineral; the magnetic portion drops behind into the shoot, the remainder falls on to a baffle board that guides it past the second magnet where more magnetic material is taken out, and so on until the tailings drop from below the last magnet, having been practically freed from all magnetic material.

At Edison[1], New Jersey, U.S.A., very low grade magnetite was concentrated for some time; the crude ore crushed to 0·06 inch mesh goes to a bank of three magnets with cast iron cores 12 ins. high, 54 ins. wide and 4 ins. thick; each bank receives a current of 15 amperes at 80 volts, the lowest magnet in each bank being the most powerful; the tailings from the topmost magnet in the bank contain 7 per cent. of iron, from the second 2·5 per cent. and from the last 1 per cent.; the concentrates contain 40 per cent. of iron. Each bank is intended to treat 5 tons of crude ore per hour. The crude concentrates are next dried and crushed to pass an 0·02 inch mesh screen, and then go to a bank of 8 inch magnets; there are three of these in a bank arranged like the 12 inch magnets. The dimensions of the cores are 8 ins. high, 54 ins. wide and 3 ins. thick, and each bank receives a current of 10 amperes at 120 volts. Such a bank will treat about 4 tons of crude concentrate per hour; the products are concentrates with 60 per cent. of iron and worthless tailings with about 1 per cent.

The concentrates are next cleaned by an air blast which carries off the finer portions, these being notably rich in apatite, this mineral being very brittle and crushing to extremely fine dust. The concentrates contain 64 per cent. of iron and pass to a third bank of magnets, there being 5 in each bank; they are 4 ins. high, 54 ins. wide and 2 ins. thick. They receive altogether a current of 17 amperes at 100 volts. In this bank it is the concentrates from each magnet that pass to the next lower ones, so that here the lowest magnet is the least powerful. The products are barren tailings, middlings for retreatment, and finished concentrates with 68 per cent. of iron. The fine dust is sent to one single magnet of the same dimensions as those of the last bank, but receiving a very powerful current, which produces concentrates like the last bank. The process was technically successful, but was economically unprofitable, the crude ore being too low-grade.

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[1] *Mineral Industry*, Vol. VI. 1897, p. 712.

B. *Wet-Working Magnetic Concentrators.*

Here again we may distinguish between machines that use electromagnets with fixed poles and those where the poles themselves move.

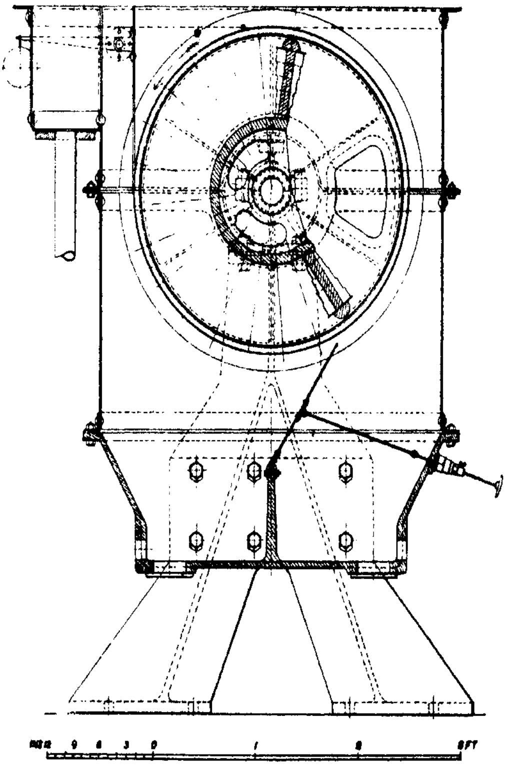

Fig. 317. Monarch wet separator. Vertical section.

Generally speaking the latter class seems to be the more widely adopted, though some may be said to combine both systems. The greatest success seems to attend those machines that draw the magnetic particles through a stream of water flowing in the opposite direction to the motion of the particles, as by this arrangement non-magnetic particles entangled amongst the magnetic ones are more readily washed out.

One of the simplest forms is merely a modification of the **Monarch Drum Concentrator** described above (page 402), its arrangement being shewn in Fig. 317; the ordinary machine was in use at Herräng in Sweden, but was replaced in the year 1899[1] by a wet drum concentrator that gave more satisfactory results. This machine has only one drum, and the electro-magnets occupy about 200° of its circumference. The drum revolves in water and the tailings run off from the front of the drum, whilst the magnetic particles, travelling for a considerable distance through the water and thus thoroughly

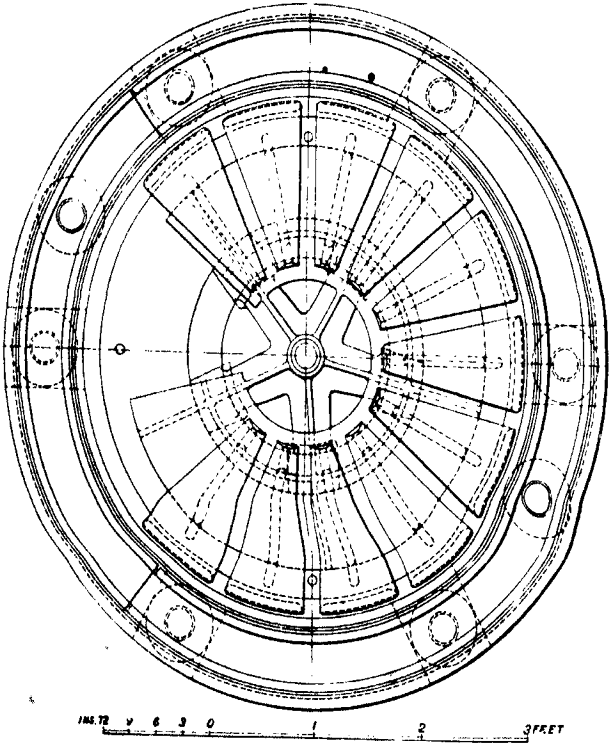

Fig. 318. Fröding separator. Plan.

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[1] *Jernkontorets Annaler*, 1901, p. 13; 1903, p. 257.

washed, are discharged at the back side. The drum makes only 8 to 10 revolutions per minute. This machine can treat about 1½ tons per hour.

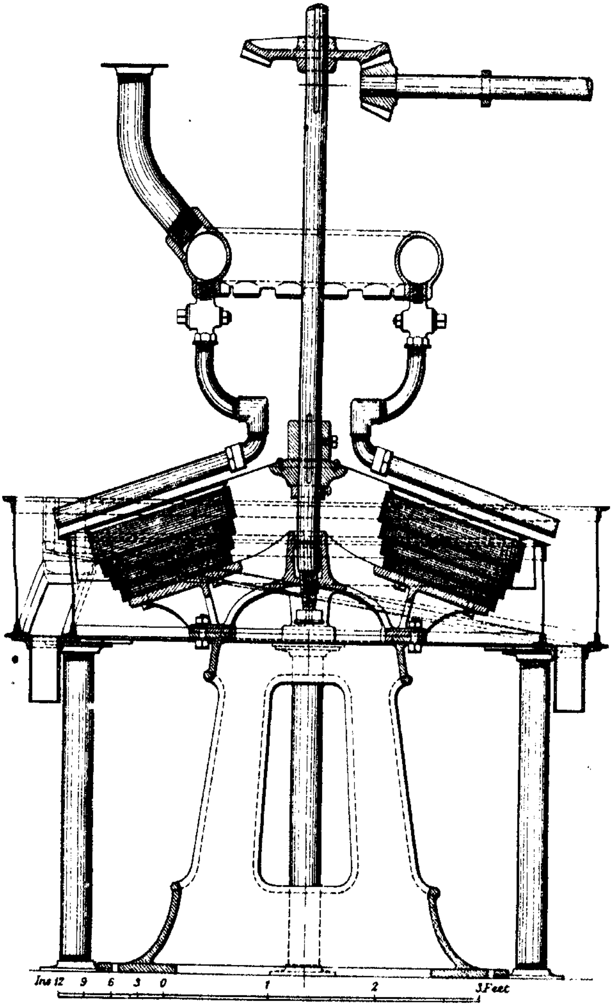

Fig. 318^a. Fröding separator. Vertical section.

The **Fröding** separator[1] consists of a slightly conical table of brass about 4 ft. 6 ins. in diameter, keyed to a vertical shaft, which is rotated by bevel gearing. Beneath this table are arranged electro-magnets, the latest form of which is shewn in Figs. 318 and 318^a; they are arranged radially, about 2 inches apart, and are of alternating polarity. As shewn, they occupy about 300° of the entire circumference of the table. Some 12 water pipes play on the surface of the table, on to the central portion of which the pulp to be concentrated is fed near to one edge of the system of magnets. As the table rotates, the magnetic portion is retained on it by the action of the magnets, whilst the non-magnetic portions are washed off by the jets of water. Finally the former particles reach the non-magnetic section and are there in their turn washed off by a jet of water. The particles are received in an annular launder surrounding the table, the portion corresponding to the non-magnetic section of the table being separated from the remainder by partitions. The current required is about 10 amperes at 110 volts, and ¼ H.P. is required to rotate the table; the water consumption is 45 to 50 gallons per minute and its capacity up to 2 tons per hour. An earlier type of this machine, shewn in Fig. 319, had three concentric rows of narrower electro-magnets[2], but gave results inferior to the later type. Fröding's separator was used for a time at Herräng.

The **Heberle** separator is shewn in Figs. 320 and 320^a[3]. It consists of a vertical tank about 3 ft. high by 20 ins. wide, within which revolves an endless rubber belt, travelling downwards on the feed-side of the tank; a series of electro-magnets is arranged along this side of the belt and inside it. The pulp to be treated is fed in through the trough *c* and runs down along the belt *e* through the space *f*; the magnetic portion adheres to the belt under the action of the electro-magnets *c* and *b*, and is carried by it into the compartment *h*, whilst the non-magnetic portion drops into *i*; the respective pulps are drawn off through pipes *g* and *k* connected with these two compartments, a flow of water being maintained through the pipes *l* and *m*, sufficient to keep the tank full. This machine can treat about 1½ tons of mineral per hour, and is used a good deal in Sweden.

The **Conkling** wet separator is somewhat like the last, but has the belt set at an angle and moving in the opposite direction.

Fröding[4] has also devised a separator consisting of a band moving

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[1] *Jernkontorets Annaler*, 1903, p. 265.

[2] Swedish patent 13349, 1900.

[3] Swedish patent specification, No. 7227, 1895; *Jernkontorets Annaler*, 1903, p. 262.

[4] Swedish patent specification, No. 20378, 1905.

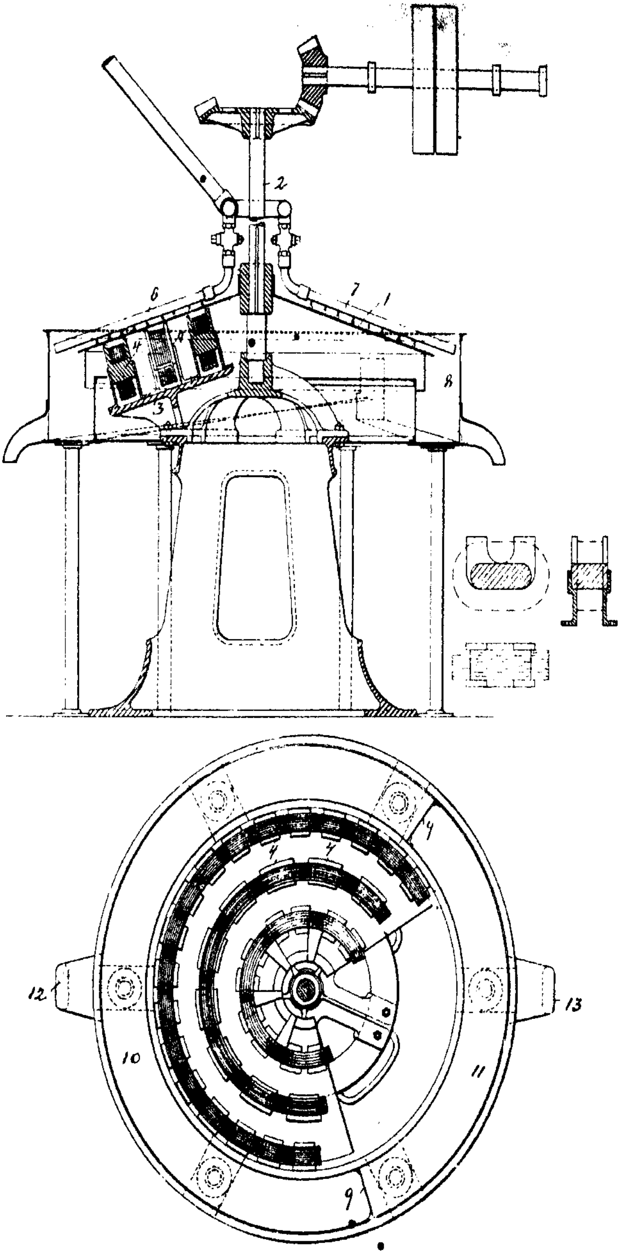

Fig. 319. Fröding separator. Earlier type. Plan and elevation.

upwards over electro-magnets at a flat angle, whilst pulp streams down it in the opposite direction.

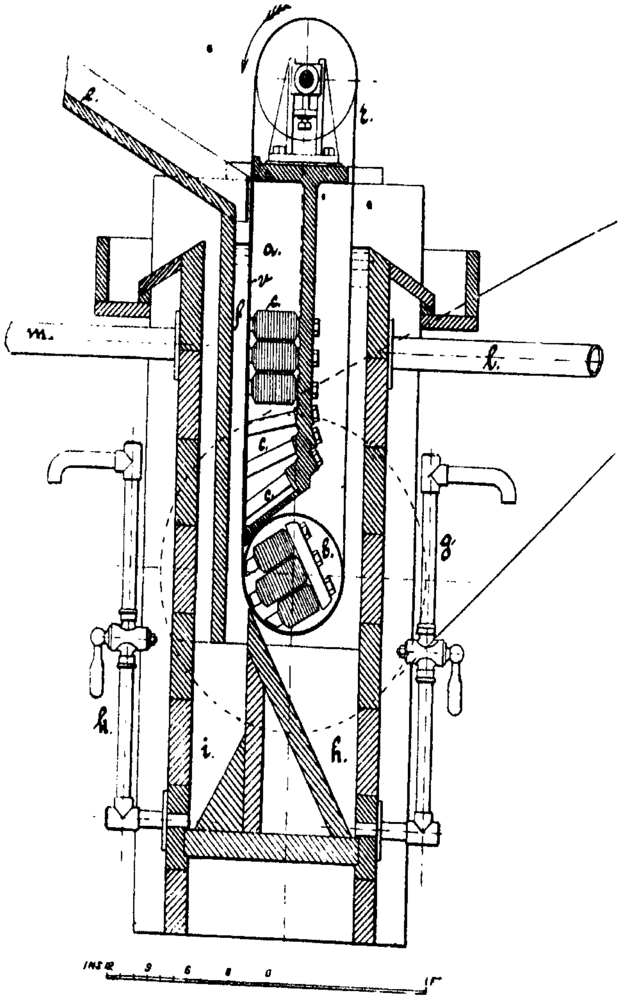

Fig. 320. Heberle separator. Longitudinal section.

The **Chase Magnetic Separator**[1], Fig. 321, is stated by the inventor to be a development of the Lovett-Finney separator. As shewn in the section, it consists of a wooden tank in which the magnetic appliance is arranged horizontally. The ore to be treated is contained in the hopper *H*, whence

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. XXI. 1892-3, p. 503.

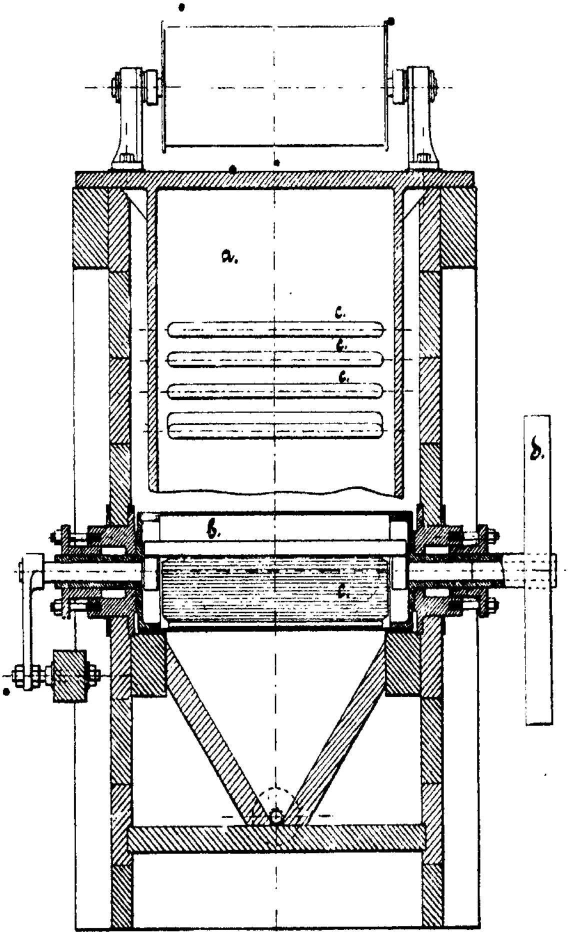

Fig. 320ᵃ. Heberle separator. Transverse section.

a feed roller delivers it on to a horizontal belt that carries it towards a magnetic roller *A*. This roller is about 4 inches in diameter and 3 feet long, and consists of a solid roller of soft iron in which two helical grooves about 1 inch square in section are cut, and in which insulated copper ribbon is wound. This makes the bar a continuous electro-magnet, and although it revolves with the belt its action is practically that of a stationary magnetic pole. The non-magnetic portions of the ore drop off at *A* into the tailings hopper, whilst the magnetic portions cling to the lower

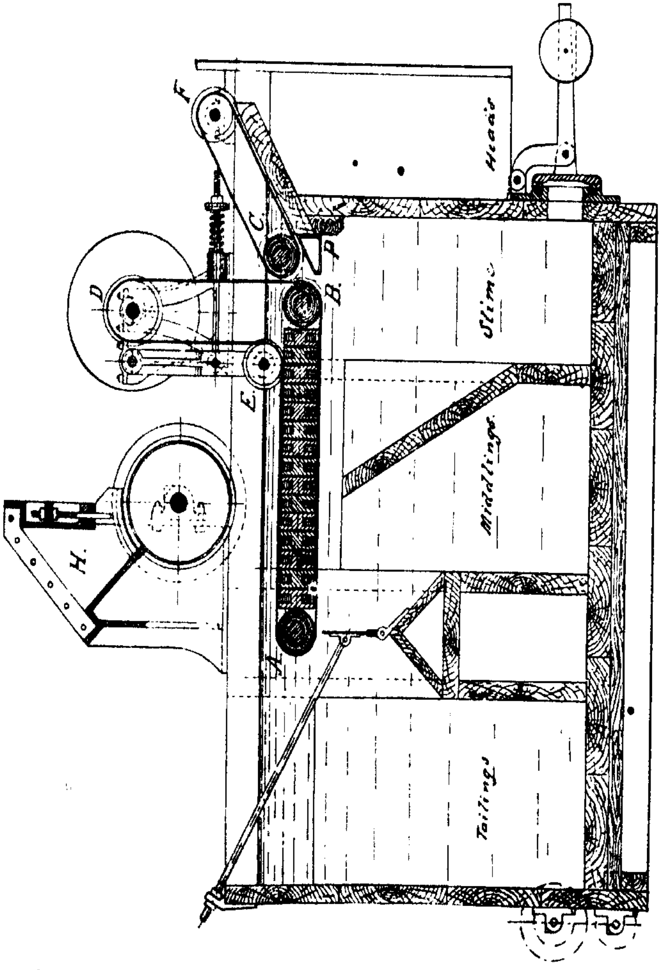

Fig. 321. Chase separator. Vertical section.

surface of the belt, which next passes underneath a series of electro-magnets with alternate polarity; during this travel the middlings and the fine dust are removed, and the magnetite is brought within the field of a second magnetic roller *B* similar to the first; from this they are taken off by the belt, passing round the third similar roller *C*, which finally discharges the concentrates over the non-magnetic roller *F*. All the separation thus takes place under water.

Eriksson's magnetic separator[1] is a good example of separators with

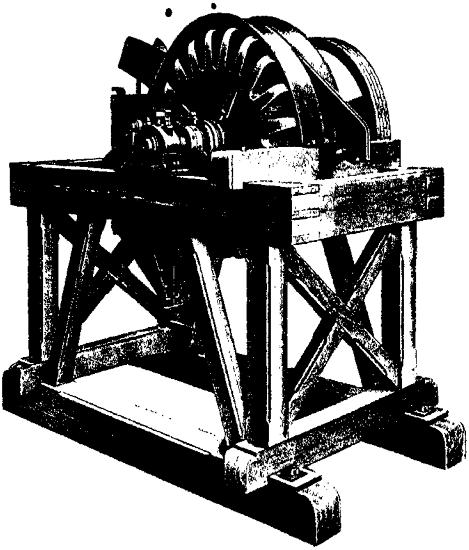

Fig. 322. Eriksson separator. Perspective.

movable pole-pieces. It is shewn in perspective in Fig. 322, and in side elevation, plan and cross-section in Fig. 323, as made in Germany by the Fried. Krupp Grusonwerk Co.; the Swedish construction differs only in some minor details. It consists of a narrow tank, triangular in side elevation, only some 3 inches in width, into which the pulp to be treated is fed by means of a hopper, this tank being kept about ⅞ full of water. On either side of the tank there are circular electro-magnets, and concentrically arranged with respect to these are two hook-shaped pole-pieces, which turn in the direction indicated by the arrow, the respective

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[1] Fried. Krupp A. G. Grusonwerk. *Jernkontorets Annaler*, 1903, p. 268.

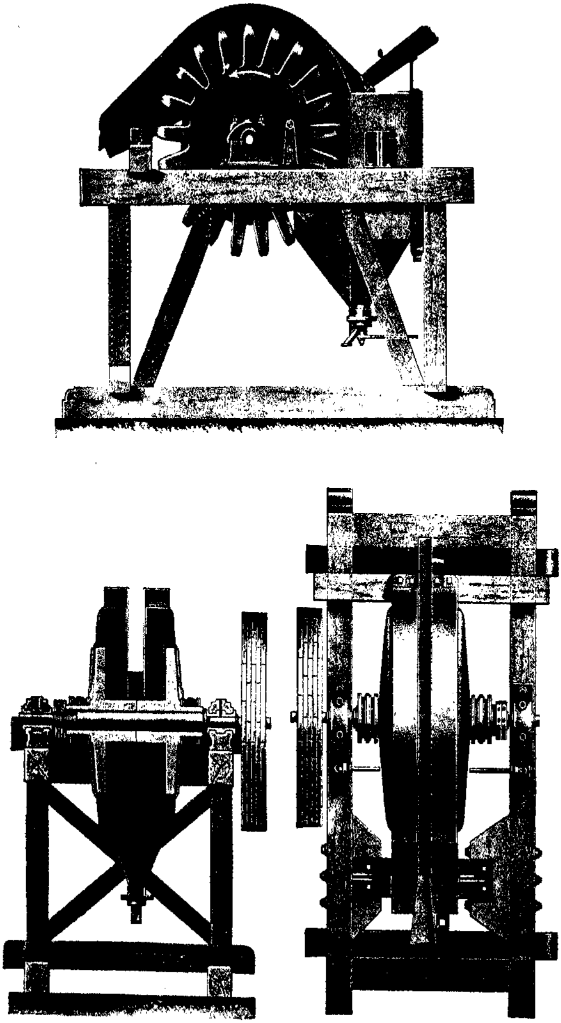

Fig. 323. Eriksson separator. Plan, side elevation and cross section.

poles on either side being of opposite polarity and those on one side exactly opposite to those on the other. When these poles revolve, the magnetic material in the pulp forms "bridges" between the opposite poles and it is thus carried with the revolving pole-pieces through the water, where it is effectively washed, and, then passing into the air, continues its course until it is deposited in a narrow trough above the tank, from which it is washed out by a stream of water. The non-magnetic portion accumulates in the lower portion of the V-shaped tank, and is allowed to escape from time to time by means of a valve at the bottom, the motion of which is controlled by a float, so that it only opens when the water in the tank has reached a certain level. The pole-pieces make about 10 revolutions per minute; the driving power required is about ½ H.P. and the electric current required is 15 amperes at 110 volts; the capacity of the machine is about 2 tons per hour. It has been used with success at Grängesberg in Sweden.

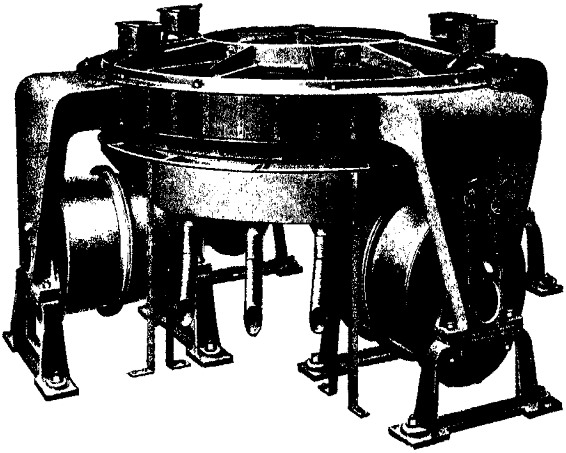

Fig. 324. Forsgren separator. Perspective.

Forsgren's magnetic separator[1], shewn in perspective in Fig. 324, and in vertical section and plan in Fig. 325, as made by the Fried. Krupp Grusonwerk Co., consists of a pair of rings of brass connected by numerous cross pieces and keyed to a central vertical shaft, thus forming a

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[1] Fried. Krupp A. G. Grusonwerk. Pamphlet.

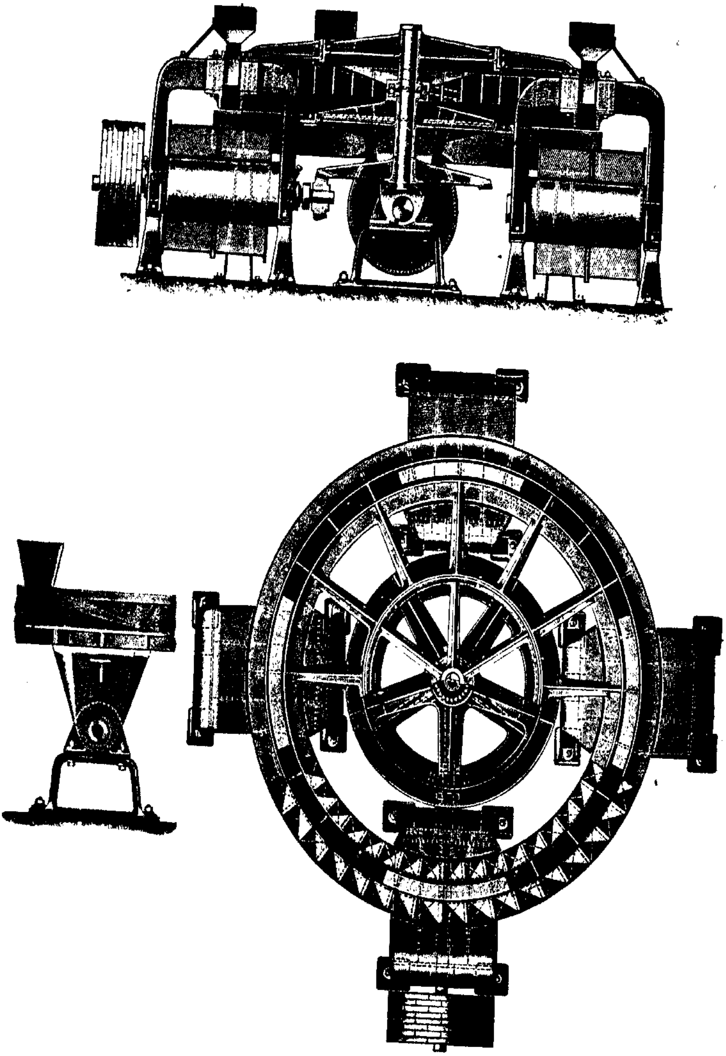

Fig. 325. Forsgren separator. Plan [illegible] erse sections.

wheel with a periphery pierced by numerous vertical slots. To the outside of this periphery are attached numerous wedge-shaped pole-pieces of soft iron, which in their rotation pass between the poles of several (4 in the figures) powerful electro-magnets. Above each electro-magnet is a hopper from which ore-pulp flows on to the open space between the inner and outer rings that form the wheel. The non-magnetic material flows straight through into troughs placed below the wheel; the magnetic portion forms "bridges" between the magnetised pole-pieces. As the latter in their revolution recede from the electro-magnets their magnetism is weakened and they deposit first the middlings and finally the concentrates, each class being washed off by streams of water into their respective troughs. This machine can be used for moderately coarse material. The ring carrying the pole-pieces makes about 10 revolutions per minute and requires about 1 H.P. to drive it. The four-pole machine will treat up to 8 tons per hour. This machine is used, like the last, at Grängesberg, on material ranging from 0·12 inch to 1 inch, whilst the Eriksson machine is used on stuff that has passed through the 0·12 inch mesh.

The **Ekman-Markman** separator consists of a hollow conical drum revolving on a horizontal axis; the drum is made of some non-magnetic material like sheet brass, and the inside is lined with a number of short studs or points of soft iron, arranged radially and embedded in cement. Outside the lower portion of the cone are placed parallel to the axis an odd number (usually 3 or 5) of electro-magnets, which induce magnetism in the iron studs as these traverse the field; when these studs pass midway between the two last magnets, which are of similar polarity, they evidently traverse a neutral zone and are thus momentarily completely demagnetised. The pulp to be treated flows through the cone, the non-magnetic portion running off at the wider end, whilst the magnetic portion is lifted up adhering to the inner lining of the cylinder, and there exposed to jets of water that wash out all the entangled non-magnetic particles. On passing through the neutral zone the magnetic mineral drops off, this action being assisted by strong jets of water, which wash it into a trough or on to a conveyor belt. It is recommended to use two such machines in series, and the current consumption of each is given as 10 amps. at 110 volts[1].

The original **Gröndal** magnetic separator[2] is shewn in Fig. 326. The inventor's description of it is as follows: A vertical shaft carries

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[1] *Glückauf*, Vol. XLIV. 1908, p. 1458.

[2] *Teknisk Tidskrift*, 1901, March 23rd.

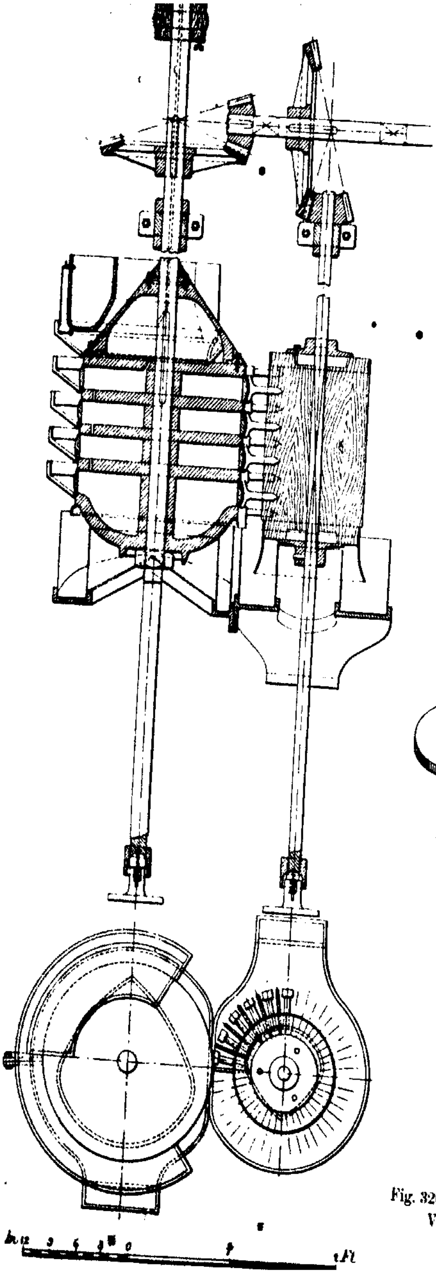

Fig. 326. Gröndal separator, type 1. Vertical section and plan.

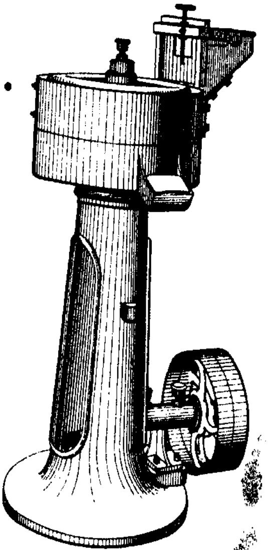

Fig. 327. Gröndal separator, type 2. Perspective.

5 discs of very soft steel set about 2¼ inches apart, between which lie coils of insulated wire, so arranged that each steel disc is more strongly magnetised than the one above it; the coils are protected by brass rings and indiarubber packing. The ore-pulp is introduced into an upper trough, which occupies about one-fourth of the entire circumference, and flows down from trough to trough, so that every portion of the pulp is brought into contact with the discs before it escapes from the machine. A plentiful stream of water is allowed to flow over the separator so as to wash off all non-magnetic particles as the apparatus revolves. The concentrates are removed by means of a wooden cylinder which carries a large number of points of soft iron, each inserted separately; they correspond to the profile of the separator, but in their nearest position are about ⅛ to ¼ inch away from it. When these points come opposite the magnetised discs magnetism is induced in the former, but as their area is so much smaller, the number of lines of force per unit area in the points is much greater; in consequence the magnetic concentrates are attracted from the discs on to these points when these are opposite the separator, but drop off as the revolution of the wooden cylinder carries the points out of the magnetic field. The concentrator spindle makes 25 revolutions, and the spindle of the wooden cylinder 225 revolutions, per minute. The current consumption is 6 amperes at 31 volts, the power consumption ¼ H.P. and the capacity of the machine is about 30 tons in 24 hours.

A second form of machine[1], invented also by Dr Gröndal, is shewn in perspective in Fig. 327 and in detail in Figs. 328 and 328^{a}. The electro-magnet consists of a spool, almost semi-circular in plan; round this rotates a brass bell, carrying a number of lamellae of soft iron. The pulp flows down from a hopper, which occupies about ⅛ of the circumference, over one end of the semi-circular electro-magnet. The magnetic particles adhere to the bell, the others being washed off; after a semi-revolution the iron lamellae lose their induced magnetism, and the magnetic particles can easily be washed off the bell. The power consumption and working capacity of this machine are about the same as that of the first type.

Gröndal's latest type of machine is shewn in Figs. 329 and 330[2], this being probably the most extensively used of any magnetic separator in Scandinavia. Fig. 329 shews a single machine in perspective with the electro-magnet inside the drum, the latter being driven by a small motor; Fig. 330 shews a duplex machine in side and end elevation, the electro-magnet proper being outside the drums and only the pole-pieces

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[1] *loc. cit.*

[2] *Journ. Iron and Steel Inst.* Vol. LXV. 1904, p. 40.

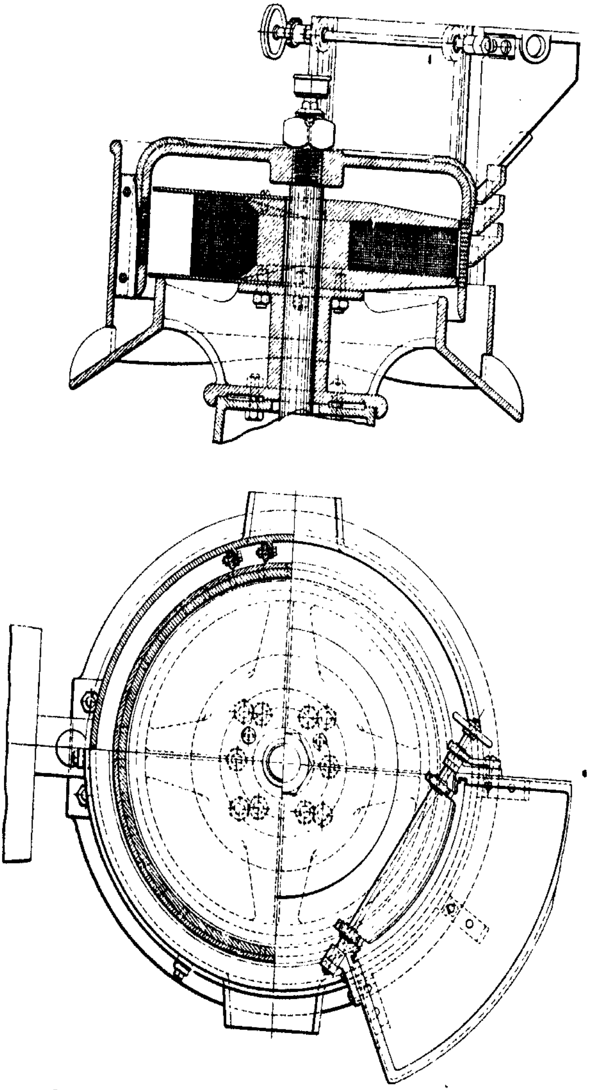

Fig. 328. Gröndal separator, type 2. Plan and vertical section.

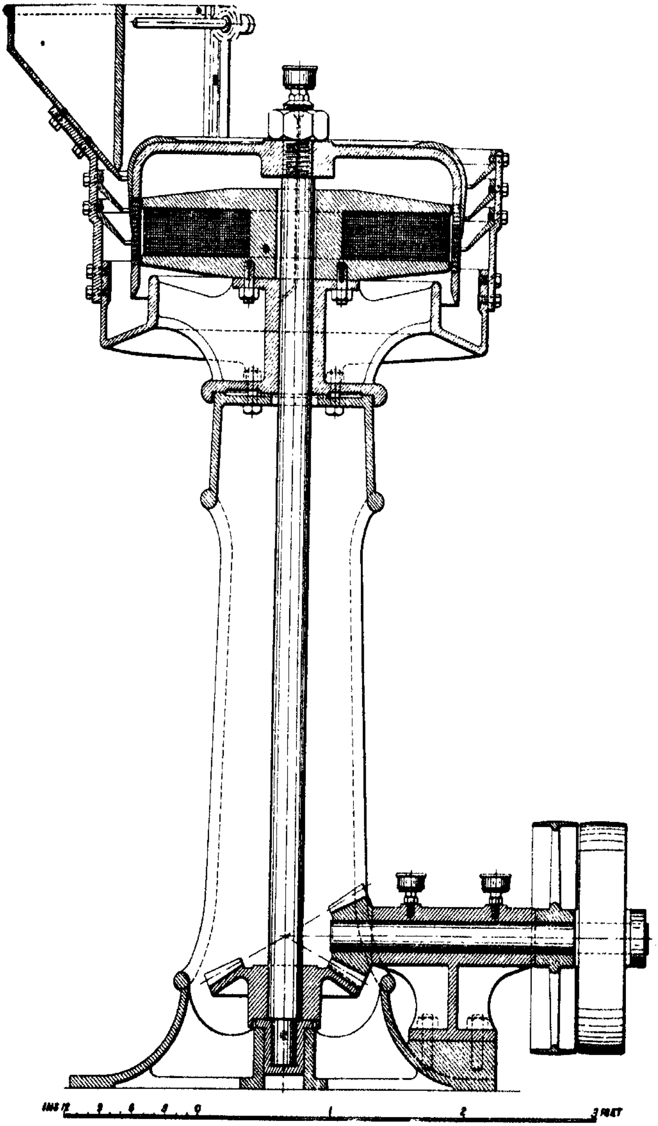

Fig. 328^a. Gröndal separator, type 2. Vertical section.

inside them, the drums being belt driven; both machines are identical in principle. The latter machine consists of a powerful cylindrical electromagnet supported horizontally, both ends terminating in massive hatchet-shaped pole-pieces *A* with the edges downwards. About each of these rotates a drum *B*, the surface of which is made up of alternate bars of brass and iron. The drum makes about 100 revolutions per minute.

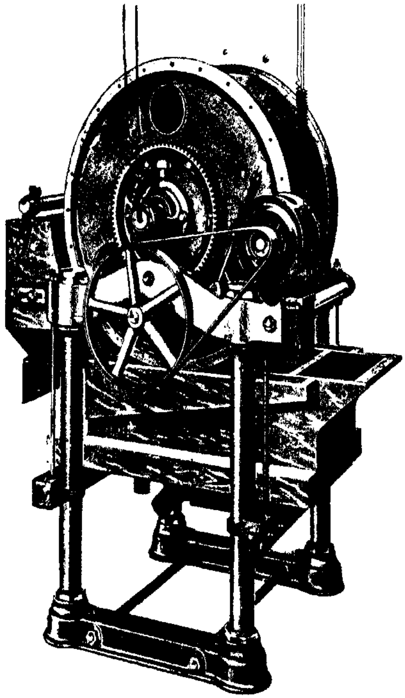

Fig. 329. Single Grondal separator, type 3. Perspective.

The ore-pulp flows through the shoot *C* into a V-shaped box divided by a vertical partition reaching nearly up to the top of the box, which is about 1 inch below the bottom of the drum. A stream of clear water entering at *D* carries all the pulp over the edge of the partition and therefore well within the magnetic field induced by the wedge-shaped poles in the rotating strips of iron (which act as secondary pole-pieces). The non-magnetic portion of the pulp flows off at the bottom of the other division of the V-shaped box, through the tube *E*. The magnetic portions adhere to the iron bars composing the drum, and the purest concentrates are thus carried to the very edge of the induced magnetic field before they are flung off by centrifugal force, aided at times by a jet of water; the

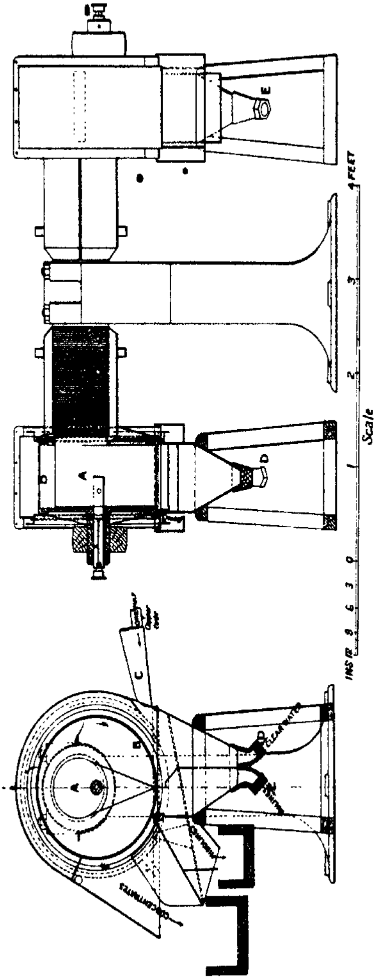

Fig. 330. Duplex Gröndal separator, type 3. Longitudinal and transverse sections.

middlings are thrown off before they reach so weak a portion of the magnetic field. Instead of the pointed pole-piece, a series of electromagnets, arranged somewhat as shewn in Fig. 304, but within the lowest portion of the drum may be used, the machine shewn in Fig. 329 being thus arranged. The principle is the same in either case, the object being the production of a powerful field at the lower front portion of the drum.

The fine slimes produced in crushing ore are apt to adhere obstinately to the concentrates, and it is an advantage to get rid of these slimes before the pulp passes to the concentrator; unless some arrangement is, however, introduced for arresting them, the slimes of the magnetic mineral may be lost with those of the non-magnetic portion. This difficulty has been overcome by the introduction of **Gröndal's**[1] **Magnetic Slime-box**, shewn in Fig. 331. It consists of a relatively narrow box, triangular in vertical section, into which the pulp flows through the launder *B*. It is met by an ascending stream of clear water from the

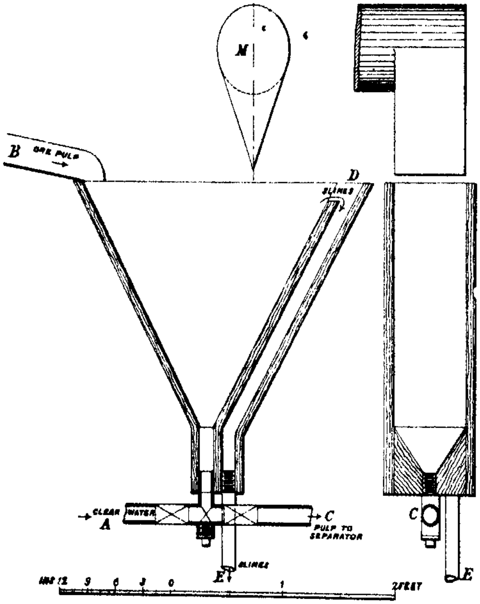

Fig. 331. Gröndal slime-box. Longitudinal and transverse sections.

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[1] *Journ. Iron and Steel Inst.* Vol. LXV. 1904, p. 40.

pipe *A*, through which the coarser particles fall and then flow off through the pipe *C* to the separator, whilst the slimes flow over the partition at *D* and off through the tube *E*. So far the appliance is merely an ordinary upward-current classifier (p. 234); above the box, however, is placed the hatchet-shaped pole-piece of an electro-magnet *M*, similar to that used in the separator but less powerfully magnetised. It is too feeble to attract the coarser particles of magnetic material, but the fine magnetic slime is arrested on its way to *D* and accumulates at the surface of the water (the magnet being unable to lift it out of the water) until it forms masses of such size as to drop down and be carried away with the coarser ore through *C*. Thus practically all the non-magnetic slimes are got rid of, whilst at the same time none of the magnetic material is lost. It would appear that magnetic particles thus agglomerated in a magnetic field retain a certain amount of magnetism and have a tendency to cohere, and are therefore less troublesome than ordinary slimes. In practice the crushed mineral goes first to this magnetic slime-box, and the pulp that escapes from *C* goes to the separators; one magnetic slime-box suffices for a double separator. Such a plant will treat from 50 to 100 tons per 24 hours, ground to about 0·02 inch; it takes a current of 6 amperes at 120 volts and ¼ H.P. to drive the separator; the water consumption is 40 to 50 gallons per minute. The single machines are made in two widths and about 24 inches in diameter; the narrow machine takes a current of 7 amps. at 110 volts, and treats 3·5 tons per hour; the double width machine takes twice as much current and will treat up to 7 tons per hour. The former requires 0·3 H.P., the latter 0·5 H.P. to drive it. These machines are often set tandem, a wide machine first to get rid of the bulk of the non-magnetic gangue, followed by a narrow machine to clean the concentrates produced by the former.

It may be pointed out that all these magnetic separators are worked with continuous currents; alternating currents can be employed, but are not advantageous, and are apt to cause undue heating of the cores and coils.

Attempts have been made to employ the rotating fields produced by polyphase currents, by means of which magnetic particles brought within such fields can be caused to travel even though the pole-pieces be stationary, so that it is possible in this way to construct magnetic separators without any moving parts whatever. Several appliances based on this principle have been proposed and some have even been tried, but up to the present none can be said to have passed beyond the experimental stage.

II. SEPARATION OF FEEBLY MAGNETIC SUBSTANCES.

It has been pointed out that the attractive power which a magnet can exert upon any body depends, other conditions remaining the same, upon the magnetic susceptibility of the body and the strength of the magnetic field. In order to attract a body of low magnetic susceptibility, a field of very great strength must be employed. This principle was first applied practically by J. P. Wetherill in the year 1896; he obtained magnetic fields of the requisite intensity by using wedge-shaped pole-pieces, and by putting these sufficiently near together. The original object of the invention was to separate franklinite and tephroite from willemite and other ores of zinc occurring at Franklin, New Jersey, U.S.A. Since then, however, the principle has found extended application, amongst the separations effected by these machines being zinc blende from galena, iron pyrites, copper pyrites, etc., spathic iron-ore from zinc blende and other minerals (dark coloured zinc blende, which contains a considerable proportion of iron sulphide, has a tolerably high magnetic susceptibility, whilst pale yellow blende is almost non-magnetic), wolfram from tinstone, quartz, etc., garnet, rhodonite, etc. from blende and galena, hornblende from apatite, limonite, haematite, and specular ore from quartz and similar minerals, dolomite from galena, etc.

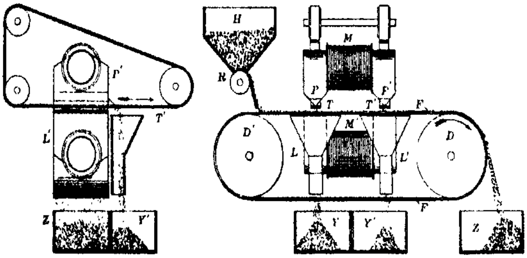

Fig. 332. Rowand-Wetherill separator. Side and end elevations.

The principle has been applied in several different ways[1]. One of the simplest and the most efficient is the "**Cross-Belt Machine**," a modification due to Mr Rowand; as shewn, in side and end elevations in

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. xxvi. 1896, p. 351; *The Mineral Industry*, Vol. x. 1901, p. 775.

Fig. 332, and in perspective in Fig. 333, a carrying belt *F* (Fig. 332) receives the ore that is fed on to it from the hopper *H* by means of the feed-roller *R*. It passes between the pole-pieces of two, or more, electro-magnets, *M*, *M′*, which have flat pole-pieces *L*, *L′*, below the belt and wedge-shaped pole-pieces *P*, *P′*, above it; owing to the intense concentration of the lines of force in the narrow edge of the upper pole-pieces, these lift even feebly magnetic particles off the belt; below each of the upper pole-pieces are travelling belts, *T*, *T′*, to the lower surfaces of which the magnetic particles adhere, and by which they are carried clear of the lower belt, until they are outside the magnetic field, when they drop into receivers *Y*, *Y′*, the non-magnetic tailings dropping into the bin *Z*. Usually two or three pairs of magnets are placed successively across the carrier belt, and as each pair is usually more powerful than the preceding one, minerals of different degrees of magnetic susceptibility can thus be removed. Fig. 333 shews such a machine with two electro-magnets. The standard machine has a belt 18 inches wide and takes ½ H.P. to drive it; it has three electro-magnets, which take respectively 5, 15 and 30 amperes at 110 volts; the capacity of the machine varies

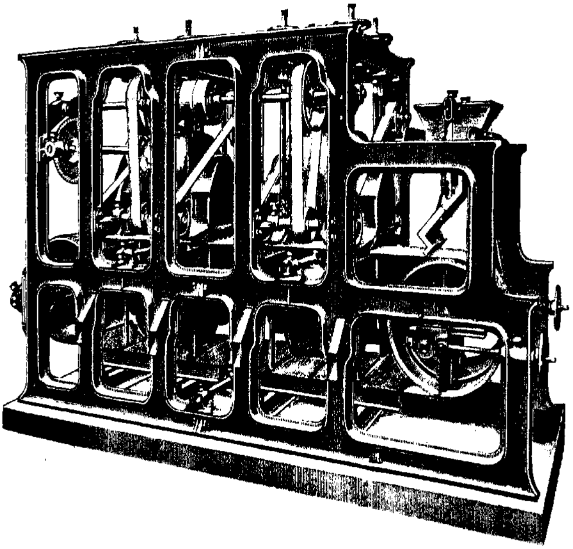

Fig. 333. Rowand-Wetherill separator. Perspective.

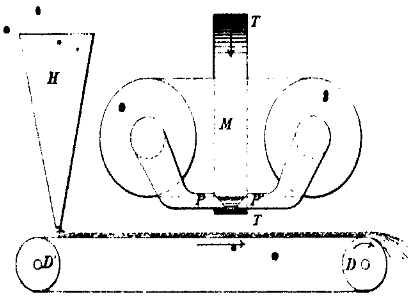

Fig. 334. Laboratory Wetherill separator. Diagram.

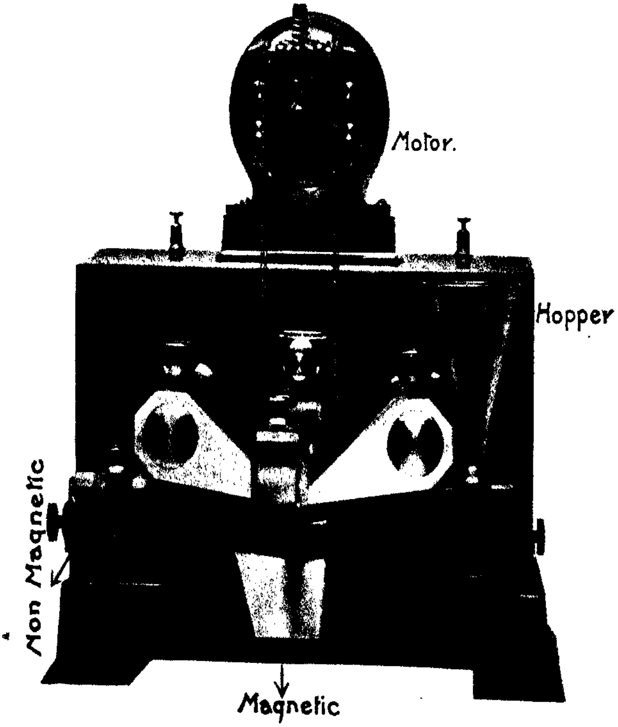

Fig. 335. Laboratory Wetherill separator. Perspective.

with the magnetic susceptibility of the mineral treated and with its coarseness or fineness from 0·3 to 6 tons per hour, being greater on coarser mineral. Its capacity is greater on more magnetic substances, the velocity of the belts being regulated by this characteristic. On ordinary ores, such as spathic ore, specular iron ore, etc., the average capacity of a 6 pole machine is 1 ton per hour, and the total power consumption 3½ H.P. At Franklin such machines have treated up to 4½ tons per hour.

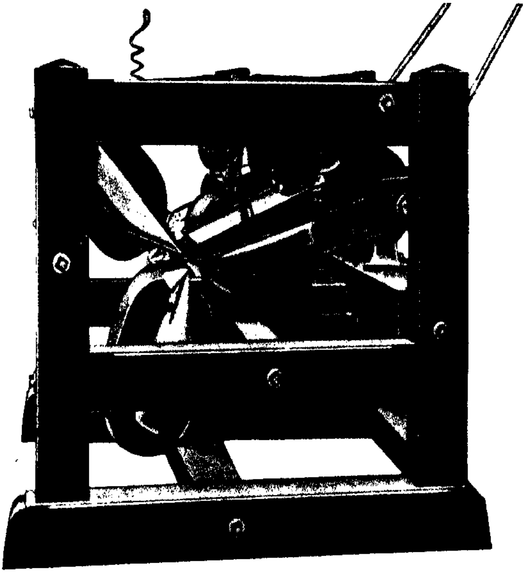

Fig. 336. Wetherill separator, type 2. Perspective.

A very convenient small machine is made on this principle for laboratory tests; this has two belts at right angles to each other, both poles of the electro-magnets being above the upper belt, as shewn in the subjoined diagram, Fig. 334; the ore to be treated flows out of the hopper *H* on to a carrying belt surrounding the pulleys *D* and *D′*; over this is supported the magnet *M* with pole-pieces *P* and *P′*; a belt *T* travels horizontally beneath the latter, and carries the magnetic particles attracted up to its underside clear of the carrying belt, when they drop into a small brass shoot. Fig. 335, from a photograph, shews a view of the complete instrument.

Another form of machine adopted to the treatment of still less magnetic minerals is shewn in Fig. 336, its working parts being shewn diagrammatically in Fig. 337. There are three electro-magnets with pole-pieces *P*, P^1 and P^2, the pole *P* being of opposite polarity to the other two. The material is fed by means of the feed-roll *R* from th hopper *H* on to the carrier belt *F*; the non-magnetic material is throw off by the momentum it has acquired, and passing over the shutter drops into the receptacle *Z*. The magnetic material being held a littl longer in the magnetic field follows the carrier belt a little way roun the edge of the pole-piece *P* and then drops into *Y*.

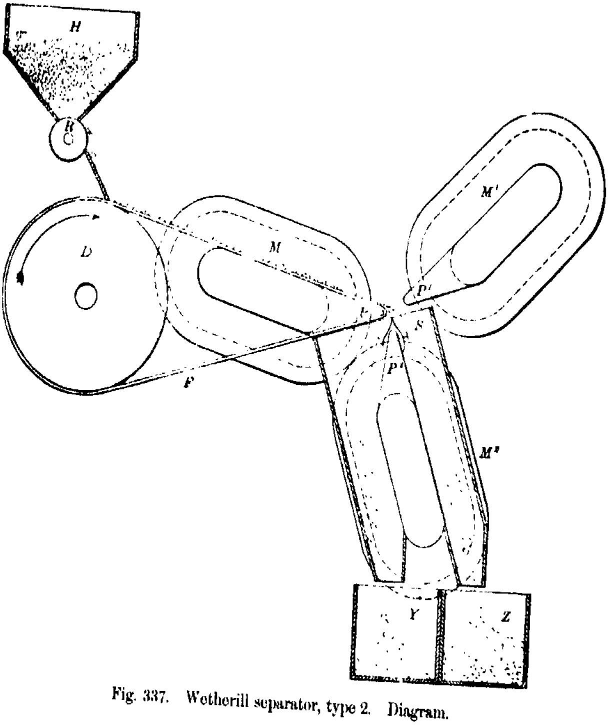

Fig. 337. Wetherill separator, type 2. Diagram.

This machine will treat material of very low magnetic sensibility, and absorbs relatively little current, as the magnetic material has only to be deflected, and not lifted. On the other hand, it is very troublesome to adjust, and there is considerable wear both on the pole-piece *P* and on the belt. It can treat about ½ ton per hour, and may be used on comparatively coarsely crushed material.

A third type is shewn in Fig. 338, a diagrammatic representation of its working parts being given in Fig. 339.

Here the mineral, fed as before by the feed roll *R* from the hopper *H* on to a carrying belt *F*, is brought within the field produced by three electro-magnets *P*, P^1 and P^2, the central pole *P* having an opposite polarity to the other two. The non-magnetic portion drops off the end of the carrying belt into the receptacle *Z*; the magnetic portion is drawn upwards into the intense magnetic field produced between the three pole-pieces, and thus against the belt *T*, by which it is carried a greater or lesser distance in the direction of the travel of the belt, so that the more magnetic portion drops into the receiver *Y*, and the less magnetic or middlings into the receiver *Y′*. The different grades are separated by means of the

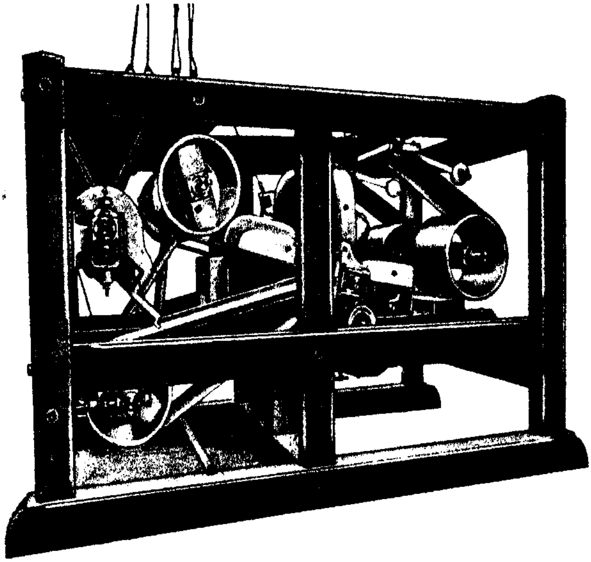

Fig. 338. Wetherill separator, type 3. Perspective.

adjustable shutters S, S'. This machine combines to some extent the principles of the two first, and is intermediate between them in its operation. It requires more electric power than the last one, but less than the first, the driving of the machine requiring only about ⅛ H.P. It can treat from 1¼ to 2½ tons of mineral per hour.

All these Wetherill machines are built by the Humboldt Machinery Company.

The **Mechernich**[1] **Magnetic Separator** is an adaption of the same principle, in which belts are replaced by rolls. It is shewn in vertical sections in two planes at right angles to each other, and in horizontal section in Figs. 340 and 340ᵃ. It consists essentially of two rollers, the axes of which are horizontal, but are not quite in the same vertical plane. The rollers are made of soft iron, and the central portion of each is wound with numerous turns of insulated copper wire, so as to convert the ends of each into pole-pieces, the upper and lower cylinder ends having opposite polarity; these ends are corrugated in the upper cylinder, smooth in the lower, which is moreover cased with stout brass or some similar non-magnetic substance. The rollers are very close together and rotate in opposite directions as indicated in Fig. 340. The ore drops out of a pair of hoppers, provided with gates capable of being closely regulated, on to the two poles formed by the ends of the lower cylinder; this acts as a feed-roll and carries the two streams of

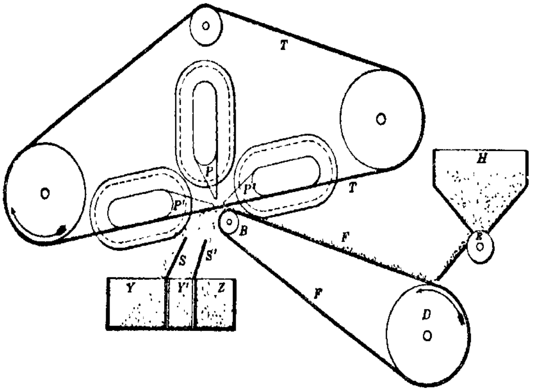

Fig. 339. Wetherill separator, type 3. Diagram.

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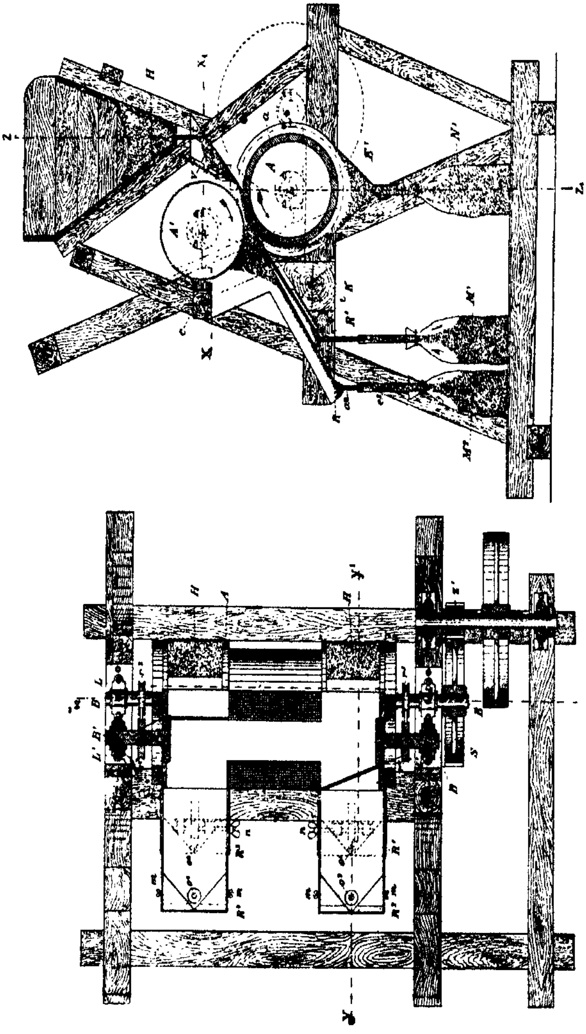
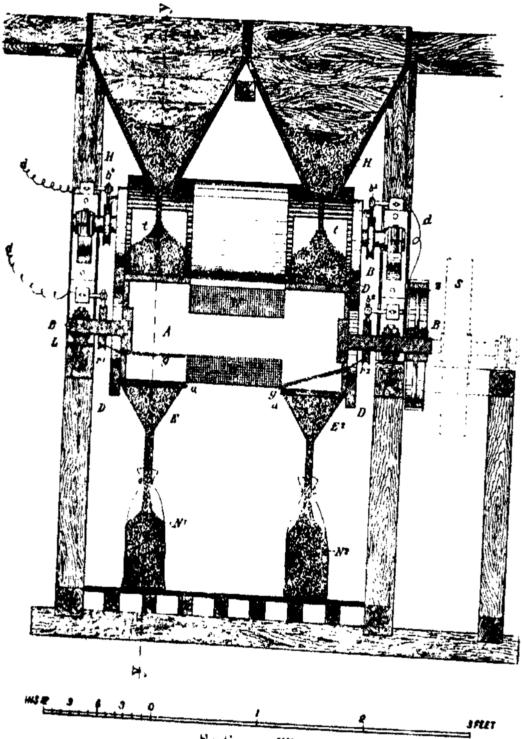
[1] *Bull. Soc. Ind. Min.* Vol. XIV. 1900, p. 1231.

ore forward until they come within the intense magnetic fields formed between the lower smooth roll, and the edges of the corrugations of the upper one. The non-magnetic portions drop straight down off the ends of the lower roll, whilst the magnetic portions are attracted towards



Section on XX. Fig. 340. Mechernich separator. Horizontal and vertical section. Section on YY.

the pole-pieces formed by the upper roll, and are carried by them to a greater or lesser distance, according as they are more or less strongly magnetic. The makers claim that their machine has many important advantages over the Wetherill machines; they state that the latter cannot exceed a belt speed of 60 feet per minute for separating rhodonite, and of 26 feet per minute for separating blende; their



Section on ZZ.

Fig. 340^a. Mechernich separator. Vertical section.

machine can use for the former a speed of rolls equal to 98 linear feet per minute. A Mechernich machine with poles of 12 inches face is said to be able to separate rhodonite alone, crushed to a sieve of 0·03 inch, at the rate of 12 to 16 cwt. per hour, and rhodonite and blende together at the rate of 6 to 8 cwt.; for the former mineral a current of 60 watts and for the latter of 200 watts is required. It may be noted that although the pole-pieces in this machine revolve, the magnetic field is stationary, and the action is therefore equivalent to that of a machine having fixed pole-pieces surrounded by a moving drum.

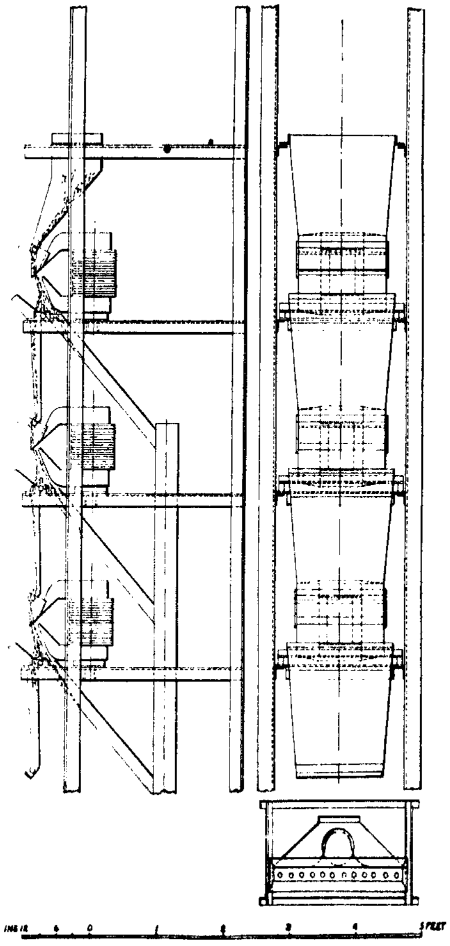


Fig. 341. Edison separator. Plan, front and side elevations.

Edison has applied his deflection principle to the treatment also of feebly magnetic bodies. The appliance and the mode of operation are exactly the same as in Fig. 316. The pole-pieces are however, as shewn in Fig. 341, much narrower, namely 1 foot 4 inches as against 4 feet, the windings are far more numerous, and the current employed is stronger. This appliance is used at Dunderland for the concentration of specular haematite; a complete bank consists of 6 magnets, and the capacity of such a bank is about 1 ton of ore per hour, the current consumption being about 10 E.H.P.

Hitherto no wet process has been devised for the separation of feebly magnetic bodies; it would seem as though the tractive force which even powerful magnets can generate in such bodies is not sufficient to overcome the resistance to their motion caused by the water. There is, however, no reason why it should not be possible to devise such machines, the deflection principle being perhaps the most promising. There is every reason to suppose that there will be a large field of useful work open to a successful wet concentrator for feebly magnetic bodies.

ELECTRICAL SEPARATION.

Differences in the electric conductivity of minerals have also been made use of to produce separation.

If a particle of mineral that is a good conductor is brought into contact with a pole highly charged with static electricity, it immediately becomes similarly charged and is repelled by the pole; in a bad conductor the process of electrification is much slower, and the mineral is accordingly not thrown off. The principle of the **Blake-Morscher** machine, which applies this property to effect separation, is shewn in Fig. 342, according to the inventors[1]. The pole consists of an insulated metal roller, rotating as shewn; an even stream of dry crushed ore is allowed to fall on to this roller from a hopper, the good conductors being thrown off from the roller and dropping into *A*, whilst the poor conductors fall almost straight down into *B*. The patentees point out that it is impossible to predict without trial whether any given mineral will be a good or a bad conductor, as there are considerable variations in the conductivity of the same mineral species; thus zinc blende from Broken Hills, Australia, can be separated by its superior conductivity from zinc blende from Joplin, Missouri, U.S.A.; among the minerals that are usually good conductors, they enumerate native metals, various metallic sulphides (excepting most zinc blendes) and graphite; amongst the bad conductors, quartz, various silicates, calcite, barytes, fluorspar, etc. One of the most usual forms of the actual machine is

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[1] The Blake Mining and Milling Co.'s Pamphlet.

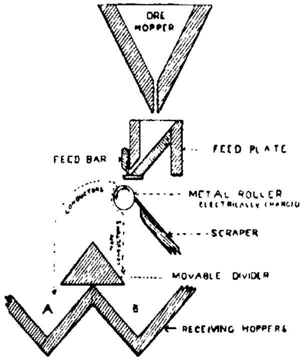

Fig. 342. Blake-Morscher separator. Diagrammatic section.

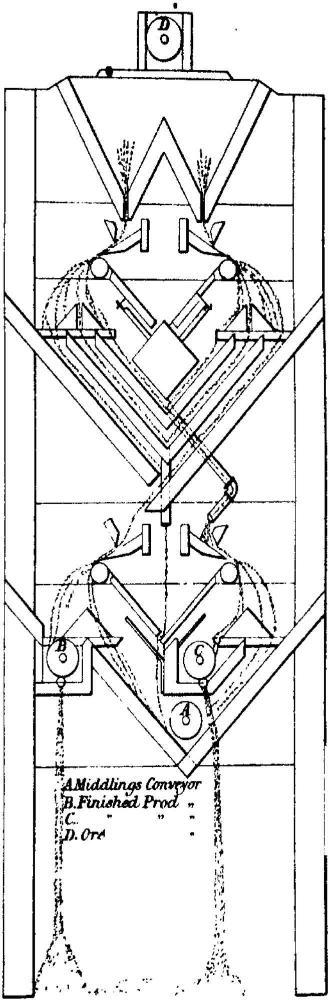

Fig. 343. Blake-Morscher separator. Elevation.

shewn diagrammatically in Fig. 343. The ore is fed by a screw conveyor *D* on to two electrified rollers which separate it into concentrates, middlings, and tailings; the concentrates and tailings each pass over another electrified roller, which separates the former into clean concentrates and middlings, and the latter into middlings and final tailings. The three classes, concentrates, middlings and tailings are removed by the respective conveyors *B*, *A* and *C*. These machines, about 12 feet long, have a nominal capacity of 12 tons (ranging from 5 to 23 tons) per 24 hours, with a power consumption of 1 H.P. They weigh about 2½ tons, and cost about £200. The electricity is generated by a frictional machine at about 20,000 volts.

The machines will work on material ranging from 6 to 100 mesh, but do best at about 20 mesh. They have been chiefly used for separating zinc blende from galena and other metallic minerals, also for concentrating molybdenite and graphite.

SEPARATION BY DIFFERENTIAL SURFACE TENSION.

Under this head may be grouped a number of methods, which appear to depend upon a difference in the adhesive power of various liquids to the surfaces of different minerals. In the absence of any experimental data on this subject, it is impossible to offer correct or satisfactory explanations of the observed phenomena; a few experiments have been made by Mr J. F. Hamilton[1], who has tabulated the power required to pull surfaces of mineral off a surface of oil under water. The tests are neither very complete nor very conclusive, but they seem to indicate that surfaces of metallic sulphides adhere to the surface of oil with greater tenacity than do surfaces of such minerals as quartz, calcite, mica-shist, etc. It seems probable that the surface tension between water and minerals such as quartz, silicate, etc., is greater than between water and metallic minerals.

Various attempts to utilise this principle were made; thus in 1886 Carrie J. Everson proposed to mix crushed mineral into a stiff paste with oil and acid, and then to wash out the gangue with water, but the process appears never to have come into use; Sutton in 1892 tried to use oil for collecting finely divided gold precipitated from a solution of auric chloride, and Robson and Cronader devised an oil concentration process in 1894, but one of the first practicable processes based on this principle was the Elmore oil process, the first patent for which is dated 1898 (No. 21948). Elmore found that if sands consisting in part of a metallic sulphide (e.g. iron or copper pyrites) and in part of a siliceous substance, such as quartz, felspar, etc., are made into a pulp with water, and this pulp is stirred up with oil, the latter will adhere firmly to the metallic minerals, but not at all to the siliceous minerals, so that if the proportion of oil be sufficient, it will form a layer above the water, carrying up with it the whole of the metallic sulphides. Subsequently he found that this so-called "selective" action of oil was in many cases promoted by the addition of a small amount of acid, an addition that was patented in 1901 (No. 6519).

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[1] *Journ. Canadian Min. Inst.*, "The Relative Attraction of some Common Minerals for Residuum Oil," Vol. VII. 1904, p. 185.

The apparatus used in practice is shewn in Fig. 344[1]. The pulp of crushed mineral suspended in water is fed into the mixer, which consists of three cylinders placed vertically one above the other; each is about 10 feet 6 inches long by 3 feet in diameter, made of galvanised iron, and having riveted to the inside a spiral blade 6 inches deep also of galvanised sheet iron. The pulp together with a due proportion of heavy mineral oil (American residuum oil, the heavy oil left behind after distilling off petroleum, is generally employed) and acid, generally sulphuric, if required, enters the upper cylinder, and the liquids are mixed by the

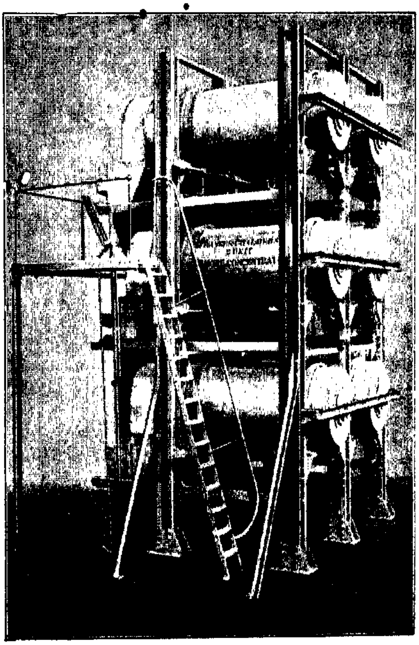

Fig. 344. Elmore oil separator. Perspective.

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[1] *Min. Ind.* Vol. XI. 1902, p. 697.

revolution of the cylinder, which, driven by gearing, makes about 6 turns per minute. The mixture is discharged into a pointed box, somewhat on the principle of the spitzkasten, from the upper part of which the charged oil flows off, whilst the pulp is drawn off from the bottom and passes to the next cylinder and thence into the third, so that the pulp is agitated three times, each time with a fresh lot of oil. The pulp then flows into a large spitzkasten, where the remainder of the oil separates out from it, and the pulp then runs to waste. The oil charged with mineral is then run into a 48 inch drum centrifugal separator, i.e. a centrifugal machine with solid walls, having a flange at the top projecting inwards. This is first filled with hot water and when revolving at a high speed (about 1000 revolutions per minute) the oil is run in. The heavier mineral passes through the wall of water and settles against the outside casing of the centrifugal, the lighter oil accumulating inside the water until it overflows at the upper edge of the centrifugal pan. The oil flows into a sump whence it is pumped back to a storage tank ready to flow down again into the mixers. The mineral accumulates until the pan is fully charged; a cover is lifted from an aperture in the bottom of the pan, and the mineral is washed out through it, and passes to a second smaller centrifugal pan, 36 inches in diameter, where a further portion of oil is removed, the concentrates being then ready for metallurgical or other treatment; they are usually sold to smelters. The concentrates are found to retain from 3 to 6 per cent. of oil. Concentrates of good grade were made by this process and in most cases a satisfactory recovery was obtained. It worked quite efficiently on most native metals and metallic sulphides, except zinc blende, which, as a rule, was found to be incapable of adhering to the oil. Obviously the process was not suitable to ores containing a large proportion of metallic sulphides of but little value, such as iron pyrites or magnetic pyrites, but on ores carrying copper pyrites, galena, cinnabar, etc., good results have been obtained. Technically the process was quite successful, but the losses of oil were found to be heavy, probably due to particles suspended in the pulp, and the process has not therefore been at all generally adopted.

The same principle has been applied in various other ways; for example, at the Kimberley diamond mines a shaking table, the surface of which was covered with thick grease, was used, it having been found that diamonds would adhere to the grease, whilst the remainder of the pulp did not.

It is also possible that the adherence of metallic sulphides to the indiarubber belt of the Frue vanner may be due in part to this same action.

A number of new processes have been devised, known as **Flotation Processes**, dating from a discovery by Mr C. V. Potter in 1901, that when the fine sands from the Broken Hill Mines in New South Wales, containing zinc blende, galena, carbonate of manganese, quartz, garnet, and other minerals, were thrown into a bath of dilute acid, minute bubbles of gas formed, which floated up practically the whole of the zinc blende to the surface, where it could easily be removed by mechanical skimmers or scrapers. This was patented as a working process (No. 1146, in 1902), the inventor's claim being for a "process of separating metals from pulverised sulphide ores, concentrates and slimes by mixing an acidulated solution therewith, stirring, heating, skimming or floating off such metals, from the surface of the whole admixture as they rise so as to recover such concentrates of metals ready for after treatment." It does not seem that the inventor really understood the cause of the effects that he had discovered, and he certainly did not appear to appreciate the part played by the carbonates and especially the rhodonite present in the ore. The real rationale of the process is still obscure; broadly speaking it appears that water adheres less firmly to zinc blende under these conditions than to the other minerals, hence the bubbles of gas are forced into closer contact with the particles of blende and cling more firmly to them than to the other minerals, thus floating them up. It was at first thought that the gas to which flotation was due was sulphuretted hydrogen, but it was soon shewn to be carbonic acid gas, evolved by the action of the dilute acid on the carbonates present, so that the process in its original form could only be applied to ores that contained such carbonates.

In 1902 Mr G. D. Delprat modified the process by using a solution of salt-cake instead of sulphuric acid, and a number of further modifications have been proposed by others, such as Lake, Bavay, Stalman-Germer[1], Cattermole, Sulman-Picard, and Elmore, in most of which the flotation effect of gas or air bubbles was increased by the introduction of some greasy or oily matter. It is quite possible that in the original Elmore oil process the buoyant effect of oil was increased by the presence of air bubbles, but these latter were only accidental and were not purposely introduced; in all cases their deliberate introduction seems to have been subsequent to the discovery of Potter's above mentioned.

It is not clear whether the Cattermole process is a flotation or an oil separation process; he patented (No. 18,589 of 1903) the agglomeration of metalliferous particles by adding 4 to 6 per cent. of their weight of oil to the pulp containing them, and also claimed (Patent No. 17,109 of

[1] *Min. Ind.* Vol. xv. 1906, p. 778

1903) the use of oleic acid instead of oil; the oleic acid could be generated by the action of mineral acid on a solution of soap.

The Sulman-Pickard-Ballot process, worked by the Minerals Separation, Ltd. (Patent No. 7803 of 1905), proposes to agitate vigorously the pulp with less than 1 per cent. of oil or oleic acid and a similar proportion of mineral acid in a special mixer, when the metalliferous minerals rise in the form of a froth or scum and can be separated in a spitzkasten. The process has hardly passed beyond the experimental stage.

The Lake process, due to Mr Alcide Froment of Traversella, Patent No. 12,788 of 1902, is stated by the inventor to depend on the following facts:

"1. When the natural sulphides reduced to powder are moistened by a fatty substance, they have a tendency to unite in spherules and to float upon the surface of water.

2. This tendency is simply retarded by the specific weight, and opposed by the gangue which imprisons the moistened sulphides in its pulverulent mass.

3. If a gas of any kind is liberated in this mass, the bubbles of the gas become covered with an envelope of sulphides and thus rise readily to the surface the liquid where they form a kind of metallic magma.

4. The formation of these metallic spherules is singularly active if the gas is in a nascent state."

The inventor has stated the principles of his process clearly enough, though he does not seem to have appreciated how largely the effects are due to surface tension. It is apparently because the adhesion of water to the gangue is stronger than to the metallic sulphides that the bubbles of gas adhere more firmly to the latter, being as it were forced closely against them by the tension of the water. The effect of the oil seems to be to accentuate this action by increasing the repulsion between the oiled surface of the sulphides and the surrounding water, due to the surface tension of the latter. The process has not come into extended practical use.

The **Elmore vacuum process** has recently assumed considerable importance, because it was found to be applicable to a number of minerals besides zinc ores, and is, for example, at the present moment being employed to treat the slimes of copper ores. The process is a flotation process which consists in generating bubbles of gas in the pulp by exposing it to a partial vacuum, when the air dissolved in the water rises up through it in the form of bubbles; a very small proportion of oil is added to the pulp, which appears as in the Lake process, to decrease the adhesion between the metallic sulphides and the water, and also enough sulphuric acid to just render the pulp slightly acid; the action of this is obscure; it may possibly act by dissolving off any film of oxide that may have formed on the surface of the sulphides, or by neutralising any alkalinity in the water, whereas Lake uses acid to generate carbonic acid gas from the carbonates which he adds to his ores.

An external view of the essential portion of the apparatus[1] is shewn in Fig. 345, whilst a diagrammatic section of the whole plant, according to the inventor, is shewn in Fig. 346. The pulp to be treated flows into the mixer *A*, somewhat similar to the mixing cylinder used in the Elmore oil process, making 16 to 40 revolutions per minute, oil and acid being allowed to flow through regulating taps into the funnel *B*. The mixture flows into *D*, which forms a small tank above the feed pipe leading to *I*, the vacuum separator proper. This consists of a conical vessel as shewn in Figs. 345 and 346, within which the scraper *L* rotates slowly (1 to 2 turns per minute). The feed-pipe enters in the centre of the bottom of the separator; the pipe *J* communicates with an air pump which maintains a partial vacuum in the upper part of the separator; as the pulp flows up the feed-pipe and into the separating vessel, air bubbles form in it by the action of this vacuum, and the particles of metalliferous mineral, which cling to these air

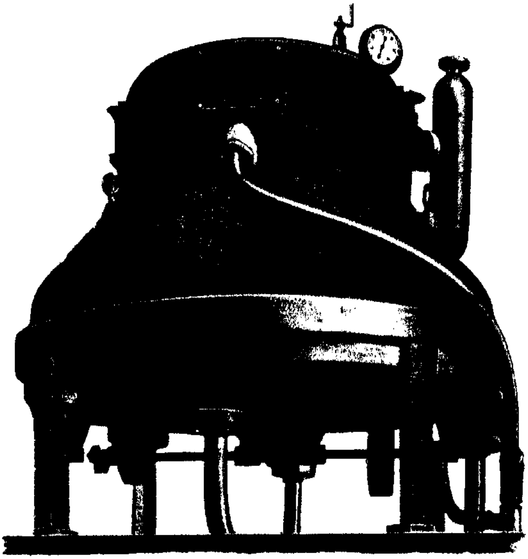

Fig. 345. Elmore vacuum separator. Perspective.

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[1] Patent specification, 17,816, 1904. Patentees' Pamphlet.

bubbles and are thus floated up by them, flow off into the collecting ring *K* and thence escape through the pipe *E*. The remainder of the pulp settles to the bottom of the separating chamber, and is carried by the action of the rotating scraper *L* to an annular groove, whence it escapes through the pipe *F*. The pipes *E* and *F* are somewhat longer than the barometric column, and thus seal off the vacuum perfectly; at the same time the tubes *E* and *F* being longer than the feed-pipe form a kind of syphon arrangement which helps to draw the pulp up into the separating chamber.

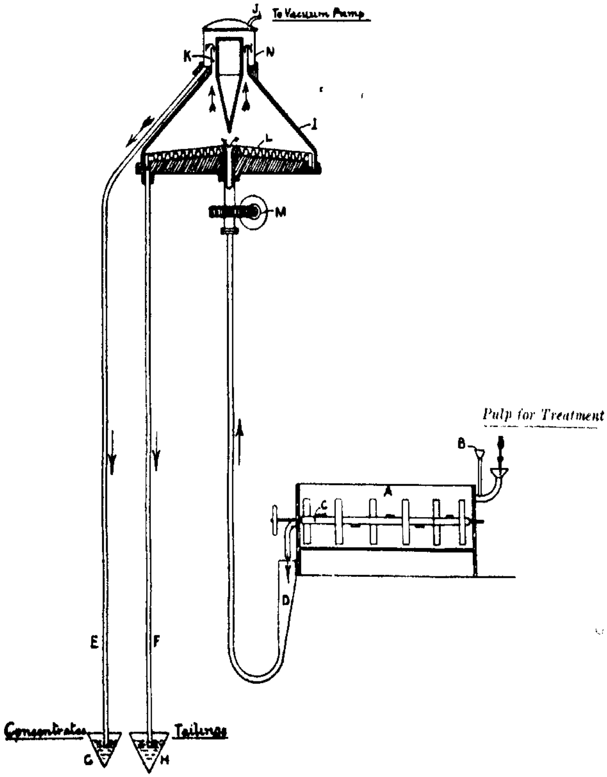

Fig. 346. Elmore vacuum plant. Diagram.

A plant has recently been installed at Dolcoath which is shewn in the accompanying drawings[1], Fig. 347; it is intended to recover the finely divided copper pyrites contained in the pulp as it comes from the stamps. This is run into a conical settling tank or spitzkasten *A*, fitted with a mechanically worked valve, opened at definite intervals, so as to allow only the sufficiently thickened pulp to run off; this flows into the mixer *B*, where it is mixed with oil and acid, and is then drawn up the tube *C* to the vacuum separator *D*; the pyritic concentrates flow off through two pipes *E* and the remainder of the pulp through the pipes *F* to a small box *G*, whence it flows to the tin-dressing plant; *H* is the air pump producing the vacuum, connected with the separator by the pipe *J*. *K* is an overflow vessel which catches any concentrates that run over, so that they may not get into the air pump. The air pump and all the plant are run by a small electric motor.

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[1] Pamphlet issued by Inventor.

In an experimental run the plant treated 1½ tons of (dry) pulp per hour, and used 3·7 lbs. of acid and 13 lbs. of oil per ton of ore; the crude pulp contained 2·41 per cent. of copper, the concentrates produced by the machine contained 17·4 per cent., and the tailings 0·23 per cent., the recovery having been from 92 to 96 per cent. of the copper present.

The apparatus is usually made with a vacuum separator 5 feet in diameter, which is considered the standard size. It requires 2 to 2½ H.P. to drive the complete unit, which can treat from 20 to 40 tons per 24 hours. The quantities of oil and acid required vary for different ores, and range from 3 to 30 lbs. per ton of ore. The process has been tried on a large number of ores with results entirely satisfactory, extractions of over 90 per cent. being frequently obtained. In regular running on the tailings from a mill dressing copper ores, which tailings contained 2 per cent. of copper, 24 per cent. of concentrates with 7·7 per cent. of copper were produced, whilst the tailings from the separator carried only 0·3 per cent. A 5 foot machine treated about 15 tons per 24 hours and used about 7 lbs. of oil and 25 lbs. of crude sulphuric acid per ton of material treated.

CHAPTER XI.

ACCESSORY APPLIANCES.

A NUMBER of accessory appliances are employed about dressing works, which, whilst not dressing appliances, strictly speaking, are yet of great importance in carrying out the work. A few of the more important of these will be briefly considered.

Transport of mineral to the works. In very many cases the dressing works are quite close to the mine, and at times the mouth of the main adit or main shaft of the mine is actually within the dressing works; in all such cases the mineral is brought into the works in mine cars; in some cases these are made side or end tipping, but, as a general rule, miners prefer a rigid car with fixed sides, and the underground roadways are rarely wide enough to admit proper tipping cars. A side tipping car of suitable pattern is shewn in Fig. 348, where the dotted lines indicate the position of the body when in ordinary use, the body being shewn

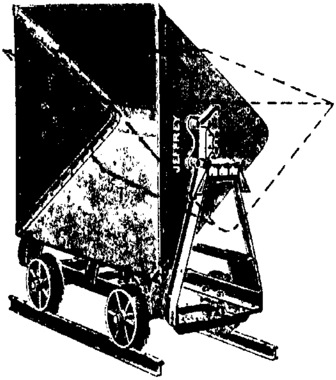

Fig. 348. Side-tipping mine car. Perspective.

as tipped. In small mines, such cars are often run on to a trestle, where the contents of the cars may be tipped out into bins or on to floors below the trestle, or else an end tipping car may be used with advantage; such a mine car is shown in Fig. 349; in this it will be seen that the wheels are set back from the tipping end, so that the weight of mineral helps to tip the load, and this plan is advantageous when it is always the same end that has to be tipped. More often, however, there are doors at both ends, and the wheels are in the middle of the body.

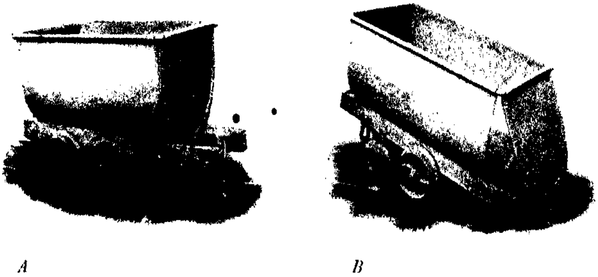

Fig. 349. End-tipping mine car; *A* running on rails, *B* tipped. Perspective.

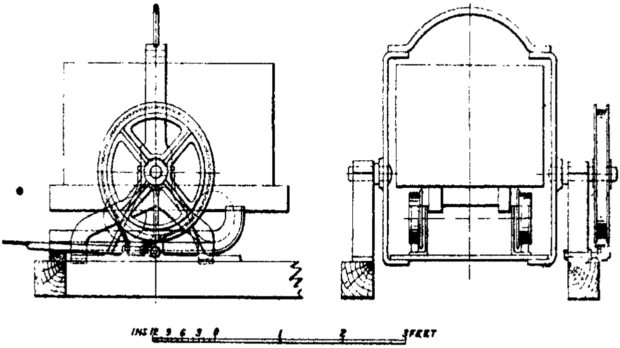

Fig. 350. Simple end-tippler. Side and end elevations.

In larger mines it is more usual to use cars with fixed sides and to use some form of "**Tippler**" or "**Tumbler**" for turning the car over and thus emptying out its contents.

Tipplers are of two kinds: end tipplers or "Kick-ups" and side tipplers.

A usual form of **End Tippler** is shewn in Fig. 350. It consists of a cage or platform hung from trunnions on either side, on to one of which is keyed a brake drum fitted with brake strap and lever worked by the foot. The rails on the floor of the tippler are turned round at the end so as to grip firmly the wheels of the mine car when run on to the tippler. The axis of suspension formed by the trunnions is so arranged that the centre of gravity of the tippler with a loaded car in position comes somewhat above it, whilst the centre of gravity falls below the axis when the car is empty. On running on a full car and releasing the brake strap, the platform with the car is easily tipped forward, whilst as soon as the load of mineral has been discharged, the tippler is easily brought back to its original position.

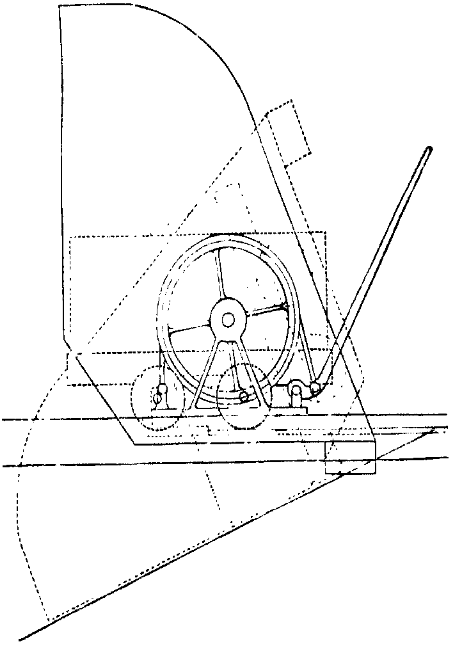

Fig. 351. Rigg's tumbler. Side elevation.

A modification of this, the usual arrangement, is known in Scotland as **Rigg's Tumbler**[1], and is shewn in Fig. 351; this is furnished with a sheet iron hood, which when the car is tipped up forms a shoot for delivering the mineral (in this case coal) smoothly and gently, and thus diminishing breakage.

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[1] *Trans. Min. Inst. Scotland*, Vol. xi. p. 164.

Another form, in which the mine car is held in place by pieces of angle iron riveted to the sides of the tippler, is **Cook's Tumbler**[1] (Fig. 352); in this the car is completely inverted when tipping takes place.

The objection to end tipplers is twofold, firstly that the mineral is thrown out roughly, and is thus, in case of a tender material like coal, apt to be a good deal broken up by its fall, and secondly that their operation is slow, because each car has to be run on to the tippler and then back again on the same road, an operation that must be performed by hand, and which takes time. In side tipplers the mineral can be discharged more gently, and the cars can be run in at one end of the tippler, tipped (usually making a complete revolution), and then, when the empty car has returned to its original position, run out at the opposite end of the tippler, so that the travel of the cars is continuously in the same direction. Furthermore such tipplers can readily be driven mechanically and may be made quite automatic in their action. There

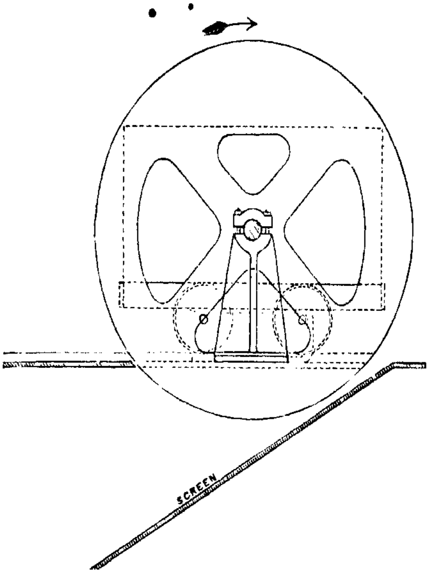

Fig. 352. Cook's tumbler. Side elevation.

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[1] *Trans. Min. Inst. Scotland*, Vol. XI. p. 180.

are very many different types, only a few of which will be referred to here.

A particularly simple form, which is intended to work like a kick-up, but to tip sideways so as to keep the cars moving only in one direction,

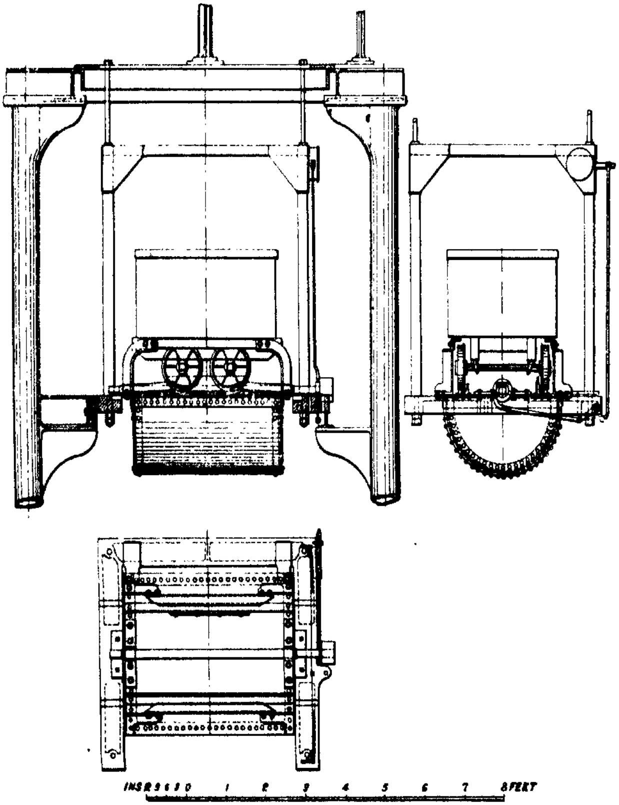

Fig. 353. Tate's tippler. Plan, side and end elevations.

is **Tate's Patent Tippler** shewn in Fig. 353. The more modern tipplers consist of cages supported on rollers, into which the car is run, and are driven by friction, by gearing or by chain drives.

A **Friction-driven Tippler**, patented by Messrs Head, Wrightson & Co., Ltd. is shewn in perspective in Fig. 354, the details being shown in Figs. 355 and 365*. It consists of a circular cage, into which the mine car is run, carried upon four rollers. One of these is the driving roller, having its tread cut out into a groove, thus: _/, into which fits a corresponding ring that forms part of the cage, this forming the friction drive. The driving roller runs loose on a shaft which is continuously driven off any convenient portion of the machinery of the heapstead. On this shaft is a sliding clutch, which can be made to engage the driving friction roller by pulling over the handle *A* (Fig. 355) of a short lever, this being done by a boy in charge of the tippler. The latter then revolves, and when it has completed its revolution, the small wedge *B* pushes back and frees the clutch and thus stops the tippler, which is further held in its exact position by the trigger *C* on the handle slipping over the pin shewn. A loaded car then runs into the tippler, the track to which is usually on a gentle gradient, and pushes out the one that has just been emptied, which continues to run forward. Sometimes the services of the attendant are dispensed with, the empty tub, as it runs out, pushing over a lever which is linked to the starting lever *A*, and thus working the tippler automatically.

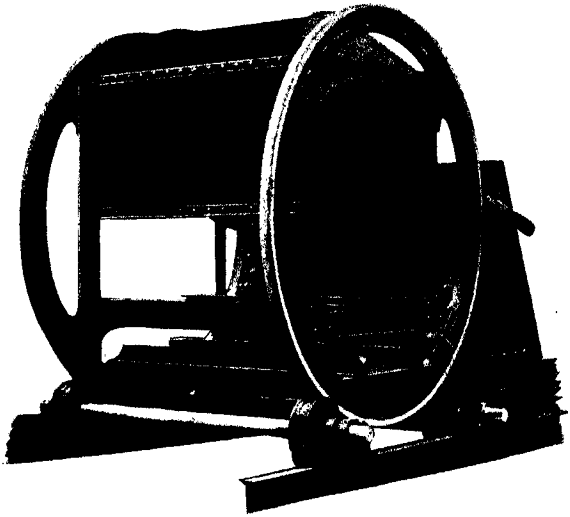

Fig. 354. Friction-driven side tippler. Perspective.

The mechanism of a gear-driven or positively-driven tippler by the

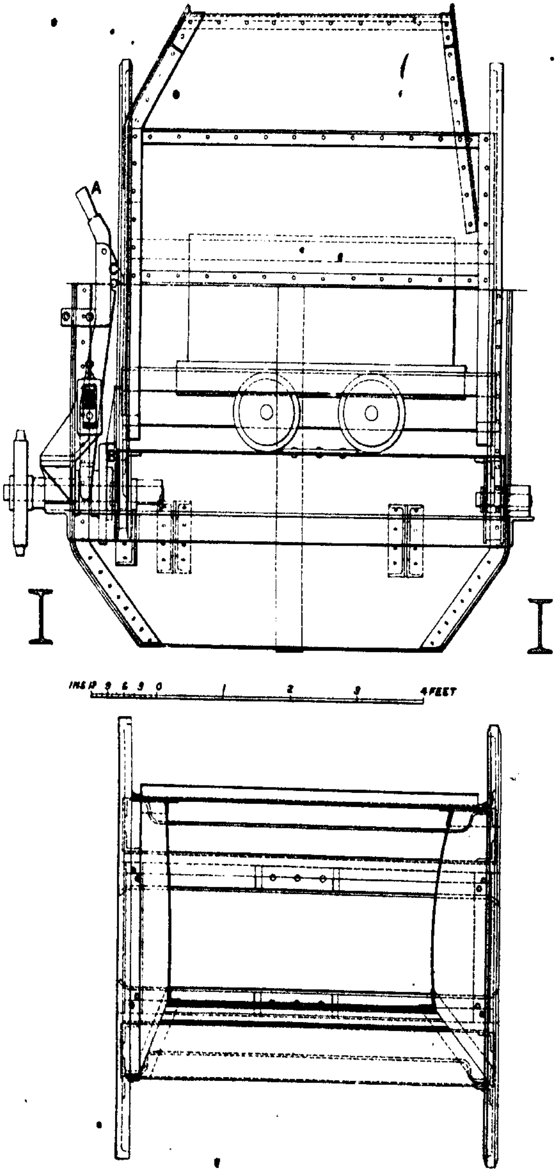

Fig. 355. Friction-driven side tippler. Plan and side elevation.

same makers, and quite identical in principle with the last named, is shewn in Fig. 356, the lettering being the same as in Fig. 355. Tipplers are sometimes made with unequal motion, so as to empty the full tub slowly and bring it back again rapidly; some are made to hold more than one tub at a time.

For **transport over moderate distances**, especially in hilly countries, wire-rope tramways may be used with much advantage for carrying the

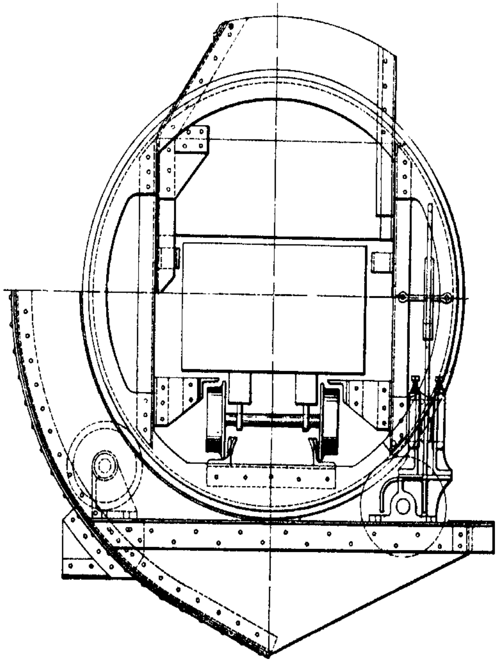

Fig. 355A. Friction-driven side tippler. End elevation.

mineral from the mine to the dressing works; there are two main types of aerial tramways, namely the double- and the single-rope. In the former there is a heavy fixed rope or rail-rope on which run sheaves or rollers from which the buckets containing mineral are hung, and they are hauled along by means of a light endless travelling rope. In the single rope system there is only one endless rope, which travels on suitable sheaves and to which the hangers carrying the buckets are clipped. The selection of the proper type for each particular case depends upon a number of circumstances; here it need only be said that the single type is usually preferred except in the case of steep gradients, long spans or heavy loads, when the double rope system, though more expensive to instal, is usually the more satisfactory. In both systems the hangers carrying the buckets run on to a length of rigid rail at both the loading and the discharging terminals, and the buckets can be tipped at any desired point along this rail, either by hand or automatically.

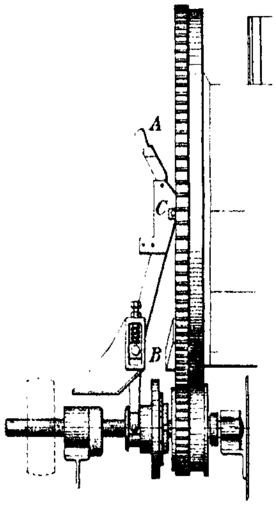

Fig. 356. Mechanism of gear-driven tippler. End elevation.

For the transport of large quantities and to long distances, mineral railways are often used. The best plan wherever practicable is to use hopper cars, and to run these on to gantries over the point where the mineral is to be deposited; usually the doors at the bottom of the cars are opened by hand, but automatic tripping devices have been used, by which the doors are made to open automatically, and sometimes whilst the train of cars is in motion. In other cases cars with movable sides or side doors have been used, so that the mineral is easily shovelled out sideways. In a few cases the cars are run on to a section of track which is swung on pivots, so that one side can be raised, often by means of hydraulic rams, until the cars are sufficiently inclined to discharge their contents.

The advantages or otherwise of any of these arrangements depend essentially upon the cost of unskilled labour; when this is cheap, discharging by hand is not infrequently more economical than any of the above automatic or labour-saving devices.

Storage of mineral. The mineral when brought to the dressing works has usually to be stored, so as to keep in hand a sufficient stock to enable the works to be kept going in case of any accident to the transport arrangements. It is usually considered that there should be sufficient storage at the works to keep these going for at least one shift, but in many cases far larger supplies of mineral are kept on hand. The necessity for this varies greatly in different circumstances and particularly depends upon the liability to interruption of the system of transport, and therefore in great measure upon the length of the latter. When the dressing works are at the very mouth of the mine, there is often no provision for storage at all.

Mineral is usually stored at the works in bins, known also as hoppers or pockets. These are of two main types. In the more usual form the bin has a lateral spout for the discharge of the mineral, in others a bottom discharge. A small bin of the former pattern is shewn in Fig. 138 as applied to a stamp mill. The latter plan is usually adopted only when very large bins are used, and these are then usually carried on arches or girders, beneath which run tracks with cars, belts, etc., for the transport of the mineral. The bottom of these hoppers is best made pyramidal, a simple sliding gate or trap door at the apex of the pyramid serving to discharge the mineral. In the more usual form the entire bin is triangular in vertical section, the angle of the bottom to the horizontal

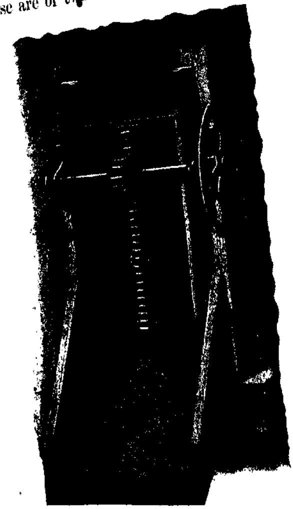

Fig. 357. Adjustable gate. Perspective.

being about 45°, whilst spouts are suitably arranged for the discharge of the mineral. These spouts communicate with the interior of the hopper by means of **Gates**, by which the discharge of mineral is controlled. It is often required to have a more or less continuous discharge, the amount of which requires regulating. In that case a sliding gate is used, the exact position of which can be adjusted by raising or lowering as required. The best arrangement is probably that shewn in Fig. 357 in which the gate is fitted with a rack in which gears a pinion actuated by a hand wheel. For wide gates a pair of racks and two pinions on the same shaft are preferable. Sometimes the gate is only a plate of stout iron, that can be raised or lowered by a chain, or that can be fixed in any desired position by means of a pin or a wedge.

Sometimes it is less important to control the rate of discharge than to let out rapidly a quantity of mineral and to stop the discharge abruptly. This may be done by having a hinged shoot, the pulling up of which against the side of the bin closes the aperture. The gate may also be a door raised and lowered by a lever, as shewn diagrammatically in Fig. 358, which enables the height of aperture to be adjusted and also to be closed rapidly at will, and which answers well for mineral broken small; for coarse pieces it is less effective, because it may come down upon a big piece, which will prevent its closing down properly and may thus leave room for smaller pieces to run out. For coarsely broken mineral, the gate is best made of an iron plate pushed up through a slot in the shoot, by means of a link and lever, as shewn in Fig. 359, which does not however provide any means of adjustment.

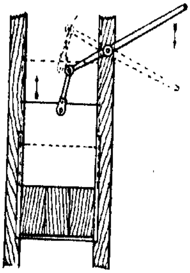
Fig. 358. Drop-door with lever. Diagram.

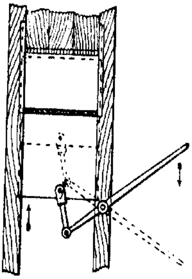
Fig. 359. Rising door with lever. Diagram.

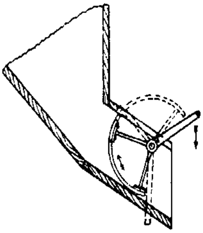
Fig. 360. Cylindrical door. Diagrammatic section.

Another satisfactory form is shewn in Fig. 360; here the gate is a portion of the surface of a cylinder which can be turned about a horizontal axis by levers as shewn; the gate may thus be opened and closed very smoothly and evenly.

A still better form of cylindrical gate, for use when an adjustable aperture is not required is shewn in Fig. 361, which gives the details of its construction. In Fig. 362 the position of the gate is shewn diagrammatically, when closing the bin at *A*, and when the aperture is open at *B*. As will be seen the gate is formed of a plate of iron about ¼ inch to ½ inch thick, the curved face of which forms the gate proper, whilst the flat portion forms a part of the shoot when the gate is open. This form is very convenient in practice.

Construction of bins. Smaller hoppers are usually built of stout boiler iron, riveted to angle iron corners; this construction is shewn in connection with rolls in Figs. 114 and 120. More rarely they are supported on wooden frames; sometimes also they are made of stout plank lined with sheet iron, occasionally of cast iron plates. Larger bins or pockets are made of almost all materials; they are at times built of masonry, or even of dressed stone, these being very satisfactory and permanent materials but high in first cost. They are mostly framed of stout timbers, and built of planks about 3 inches in thickness; such bins are shewn in connection with a stamp mill in Fig. 138; a good plan consists in making the lining in two thicknesses, an outer one say of 2 to 2½ inches and an inner one of 1 to ½ inch, the inner one being readily renewable when worn. The outer planks should run diagonally, whilst the inner ones run vertically, in the direction in which the mineral slides. Bins are often built of plank lined with iron plate ⅛ to ⅜ inch in thickness. The bottom, which is exposed to most wear, is often laid with cast iron plates. A very good arrangement consists in lining the sloping bottom with old rails, laid as closely as possible together; fine mineral soon packs into the space between the heads and a very durable lining results.

Of late years large storage bins have been built very successfully of ferro-concrete[1]; several of these have been erected in France. They may be lined with iron in quite the same way as wooden bins.

Conveyors. Mineral frequently requires to be conveyed from one part of the dressing works to another; this is sometimes combined with a certain amount of lifting, so that conveying appliances act also as elevators, and the latter term will therefore be restricted here to lifting mineral in an almost vertical direction.

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[1] *Trans. Min. Inst. Eng.* Vol. XXXIII. 1907, p. 15.

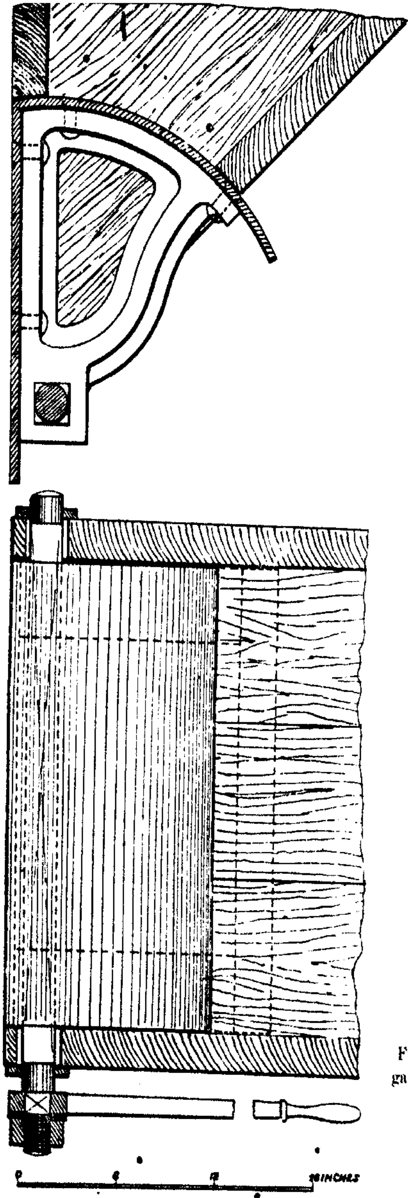

Fig. 361. Improved cylindrical gate.

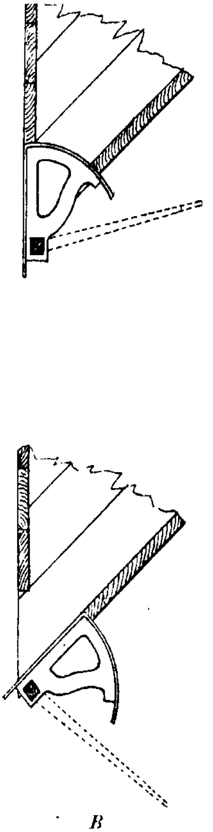

Fig. 362. Improved cylindrical gate; *A* shut; *B* open. Diagram.

Coarsely broken mineral is usually, and finely crushed mineral is sometimes, carried in cars of the same form as mine cars, an ordinary side tipping car being particularly convenient; the capacities of these vary usually between 15 and 25 cubic feet, and they run on tracks laid with light rails, the gauge being usually 18 to 30 inches. For short distances and small total quantities they are generally pushed by hand; for long distances, practically horizontal, endless wire rope haulage, or preferably perhaps endless chain haulage may be employed. For short distances, and large quantities, especially where loads have to be taken up an incline, **Creepers** are used. These consist of an endless chain running in a suitable channel in the centre of a railway track between the rails. At proper intervals pieces are secured to the chain which bear against the axles of the cars, and thus carry the latter up the incline.

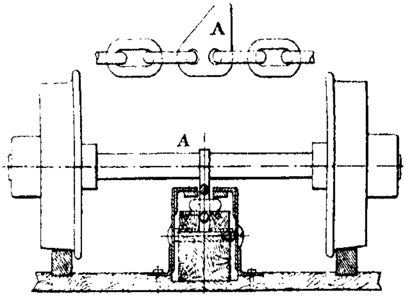

Fig. 364. Creeper chain. Side elevation and section.

These pieces automatically engage the axles of any cars run on to the foot of the incline and the cars automatically run off at the head, the track being caused to slope downwards away from the head.

Such a creeper is shewn in Fig. 363, this representing a colliery creeper for running empty tubs up an incline, the details being clearly shewn in the figure. A somewhat different form of chain is shewn in Fig. 364, in which the catching piece, *A*, is a simple triangular piece of iron plate, whilst Fig. 365 shews a flat linked chain of sufficient width to maintain a straight line, with the catching pieces, *A*, that engage the axles secured to the middle. The speed of a creeper chain must be slow, say 2 to 3 miles per hour; somewhat higher speeds may be employed by using a pushing piece with a spring buffer, but this adds complications and is not in any great favour.

All the various forms of picking belts described in Chapter III, pp. 88—100, may be used for conveying purposes, belts of the Robins pattern, p. 96, being largely used for this purpose. These belts may be up to 6 ft. in width, and will work on gradients of as much as 20°; their speed may be as much as 600 ft. per minute. The **Jeffrey Century Conveyor**, Fig. 366, differs from the Robins in that both sides of the belt are inclined, whereas in the Robins, as shewn in Fig. 85, the central portion is horizontal; the Jeffrey century conveyor uses an indiarubber belt like the Robins. Woven canvas belts have also been used successfully for conveying purposes.

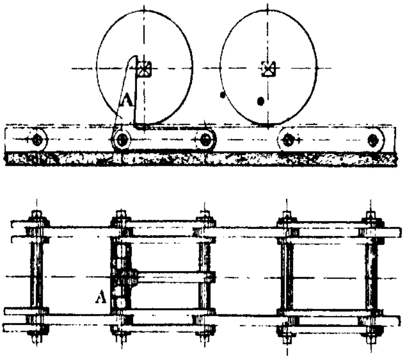

Fig. 365. Creeper chain. Side elevation and plan.

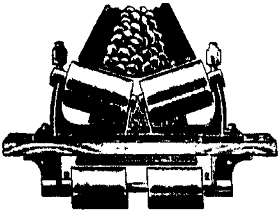

Fig. 366. Jeffrey century conveyor.

Bucket Conveyors, consisting of a series of boxes, carried on link chains fitted with rollers, are also used for moderately small stuff. Fig. 367 shews such a conveyor made by Messrs Graham, Morton & Co., Ltd.

All the swinging screens described in Chapter II are also used as swinging conveyors. The **Zimmer Conveyor** (see pp. 48 and 80) was one of the first of these and has been used largely for conveying mineral especially coal, which is moved forward at the rate of 50 to 60 feet per minute; a 36 inch trough 6 inches deep will carry up to 30 tons of coal per hour.

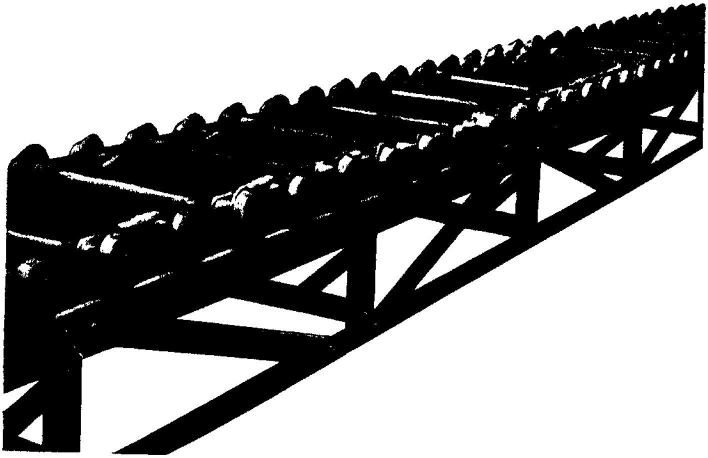

Fig. 367. Bucket conveyor. Perspective.

Swinging conveyors are not well-adapted for carrying mineral up-hill and are used especially for transport in a horizontal direction.

There are a number of appliances that may be classed as **Trough Conveyors**, in which the mineral is moved along a fixed trough by various devices.

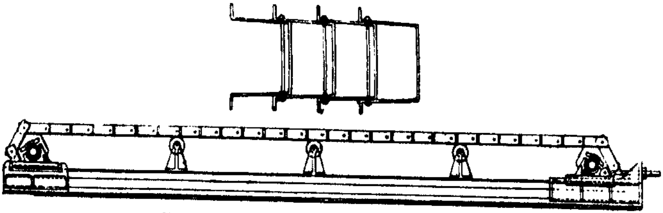

Fig. 368. Scraper conveyor. Elevation and plan.

The **Scraper Conveyor** consists of a trough in which travels an endless steel chain which drags the material along. One form, as made by Messrs Head, Wrightson & Co., Ltd., is shewn in Fig. 368.

As shewn, the chain is made of flat steel strips 4 inches deep by ⅜ inch thick bent into the shape shewn, the links being coupled together by steel pins. This form is suitable for quite coarse mineral but is best applied to a light material like coal.

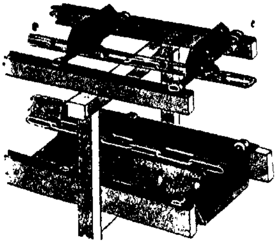

Fig. 369. Scraper conveyor. Perspective.

Another type, better suited to small stuff, is shewn in Fig. 369, this being made by the Jeffrey Manufacturing Co. Another form by the same makers also for finer material is shewn in Fig. 370.

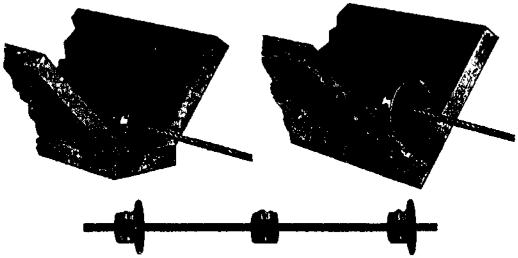

Fig. 370. Scraper conveyor. Perspective.

Finally the **Screw Conveyor**, which is very largely used for conveying finely crushed material, is shewn in perspective Fig. 371, this also being a pattern made by the above named Company. Another very similar form is shewn in plan and elevation in Fig. 372.

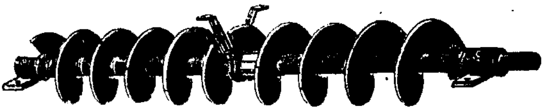

Fig. 371. Screw conveyor. Perspective.

It will readily be understood that these various conveyors are made in a large number of different types to suit special conditions; the principles should however be clearly intelligible from the illustrations here given.

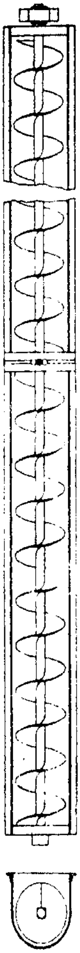

Fig. 372. Screw conveyor. Plan and elevation.

Elevators. The most usual plan for elevating very coarse stuff is to use a hoist lifting one or more cars containing ore. These hoists may be worked on any desired principle; they are often mechanically driven either off a convenient line shaft or by a special motor of any kind; in recent times electric motors have been much used for the purpose. A mechanically driven hoist of the most usual construction is shewn in Fig. 373, this being a pattern built by the Humboldt Engineering Co. Cylinder hoists, either worked direct or with rope gearing, the ram being moved either by steam or compressed air, are also much used; hydraulic lifts may be employed but are usually rather too slow for this purpose. In small works a water-balance hoist is often used especially where an ample supply of water at a high level is available.

For medium- and small-sized material the bucket elevator is in very general use. It may be vertical as shewn in Fig. 375, but is more often inclined at 10° to 20° to the vertical, as shewn in Fig. 374. It consists of chains of various types to which buckets are attached, which pick up the material delivered to the boot at the lower end of the elevator and discharge it at the upper end. The chains may be flat link chains passing over hexagonal or octagonal tumblers, or ordinary pitch chains passing over sprocket wheels. Generally the upper end is mechanically driven, whilst the bearing of the lower end runs in slides with proper tightening screws.

In many cases these elevators are made of stout canvas, indiarubber or Balata belts, to which pressed steel buckets are riveted. Such buckets are made from 4 inches to 36 inches long and 3 to 12 inches wide. With buckets say 18 by 8 inches, spaced 18 inches apart, a belt, travelling at a normal

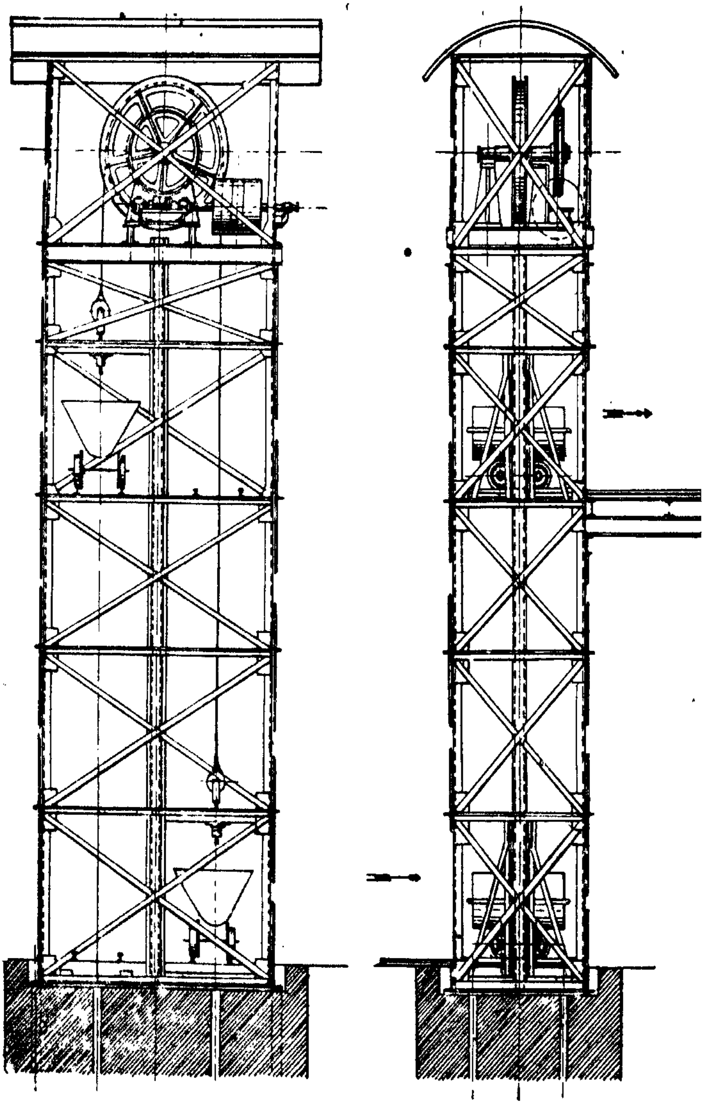

Fig. 378. Hoist for cars. Side and end elevations.

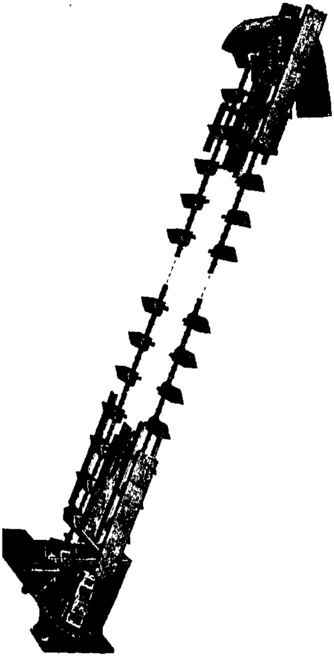

Fig. 374.
Inclined chain-bucket elevator. Perspective.

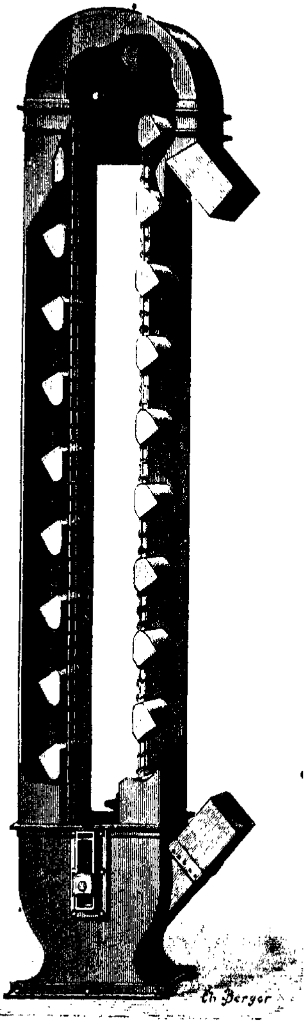

Fig. 375.
Vertical chain-bucket elevator. Perspective.

speed such as 200 feet per minute, would elevate about 3000 cubic feet of material per hour. Speeds up to 500 feet per minute may be used. These elevators are generally employed for dry material, but work equally well upon wet; in the latter case the buckets are sometimes made of punched or perforated sheet steel, so as to allow the water to drain off more or less completely whilst the material is being elevated.

The power required to work a bucket elevator may be approximately determined as follows:

Let V be the speed of travel of the chain or belt in feet per minute.

Let D be the distance apart of the buckets in feet.

Let W be the weight of material contained in each bucket.

Let H be the total height of lift.

Then the horse-power required is approximately:

$$\frac{V \, . \, W \, . \, H}{11{,}000 \, . \, D} .$$

The bucket elevator has almost displaced the Raff wheel, which was at one time largely used for the same purposes, and is still seen at times. A raff wheel is built like a water wheel with the buckets on the inside, i.e. opening inwards. This wheel is driven by gearing or belting and revolves slowly, a shoot discharging the material to be elevated into the buckets; as the wheel revolves this material is lifted and is discharged when the bucket reaches its highest position and is therefore inverted.

Conveyance of pulp. Pulp may exceptionally be carried horizontally in bucket conveyors, but is more often allowed to run down in launders inclined at such an angle as to enable the pulp to flow. These launders are made of wood, stout sheet iron or cement; the best shape for a launder is one that gives the maximum of sectional area for a given perimeter, and launders of iron or concrete are therefore usually made semi-circular. Wooden launders are usually made rectangular, and should then be so proportioned that the width is twice the depth; if they be made with the sides inclined at an angle of 60° instead of being vertical, their transporting power will be increased, but such launders are more difficult to construct and keep tight, and are therefore seldom seen. Small launders are sometimes made V-shaped in cross-section, the angle at the apex being a right angle. The rate of flow of the pulp is best determined experimentally; the well-known formulas for the flow of water in channels will give an approximation to the truth, but cannot be relied on in all cases, as the constants have only been determined for a few special cases. The following formula due to Italian engineers is sufficiently correct for most purposes:

Let u be the mean velocity of flow in feet per second.

Let r be the mean hydraulic depth in feet (i.e. the area of cross-section of the stream in square feet, divided by the perimeter of the wetted portion of the launder in feet).

Let i be the gradient of the launder, expressed as a fraction.

Then $$u = 87\sqrt{ri}.$$

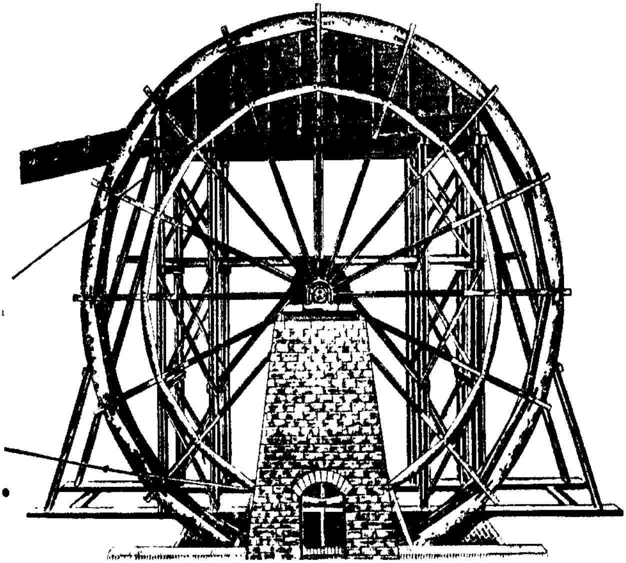

Fig. 376. Tailings wheel. Side view.

The mean velocity of a water current that will just move mineral matter of different sizes has been determined for rivers as follows:

Light clay	is just moved at a mean stream velocity of	8	inches per second
Sand	,, ,, ,, ,, ,, ,,	16	,, ,,
Small gravel	,, ,, ,, ,, ,,	32	,, ,,
Pebbles	,, ,, ,, ,, ,,	40	,, ,,
Broken stone	,, ,, ,, ,, ,,	64	,, ,,

Elevating pulp. This is often done by means of so-called sand wheels or tailings wheels, which are built at times of large dimensions;

they have been used largely on the Witwatersrand for elevating the tailings from stamp mills for further treatment, a typical example being shewn in Fig. 376. These are often made up to 50 feet in diameter, and are built exactly like raff wheels, except that they are much larger. The shrouding of these wheels is usually of wood, but its outer periphery, which forms the bottom of the buckets, is often of iron plate about ⅛ inch thick, the partitions between the buckets being of the same material. The buckets are usually about 12 inches deep and 15 to 24 inches wide, and are pitched 12 to 15 inches apart. The peripheral speed of the wheels is about 500 feet per minute.

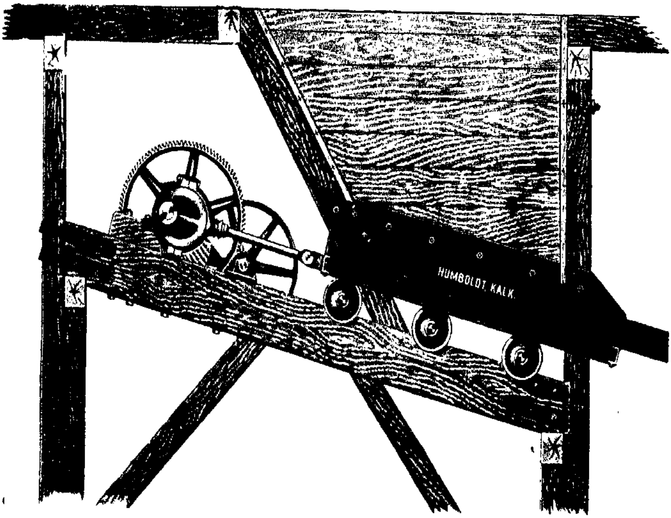

Fig. 377. Reciprocating tray feeder. Sectional elevation.

For high lifts pumps are employed, the plunger type being the most generally satisfactory; centrifugal pumps and the spiral sand pump are also used for moderate lifts. The ordinary bucket elevator can be and often is used for elevating pulp; the buckets are then often of very large size.

Feeders. It is very important in many cases that the material to be treated shall be supplied regularly and uniformly, more especially to crushing machinery; appliances for effecting this are known as feeders, and are made on a number of different principles.

The **Reciprocating tray feeder** consists of a tray set at an angle of about 15°, and forming the bottom of a hopper. It is shewn in Fig. 377 as built by the Humboldt Engineering Co. It is moved to and fro by an eccentric or a crank, at some 5 to 15 strokes per minute; on the back

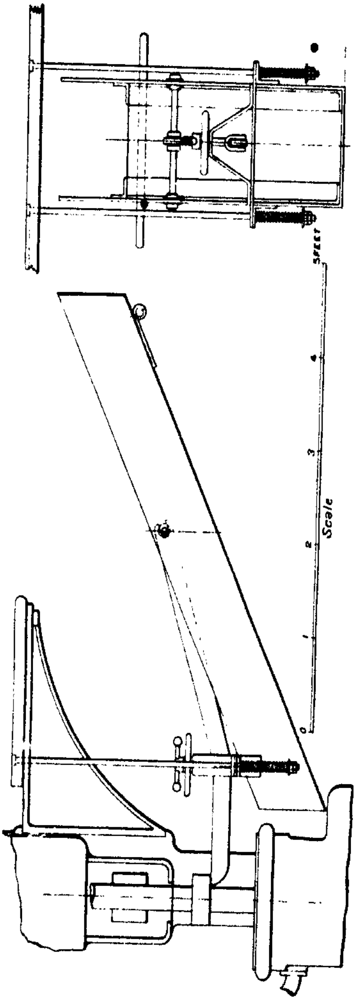

Fig. 378. Bumping tray feeder. Side and end elevations

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stroke the material drops out, whilst the forward stroke advances the tray to receive a fresh portion. This feeder is very well adapted for feeding rock-breakers, especially the secondary rock-breakers in tandem breaking.

There are numerous **Shaking** or **Bumping tray feeders.** A very simple form as applied to stamp mills is shewn in Fig. 378, which consists simply of a shoot held up by iron bars supported on strong spiral springs; the ore runs out of the hopper on to this shoot, which is at an angle flatter than the angle of repose. The middle stamp stem of the battery has a collar attached to it, at such a height that when there is no ore on the die this collar strikes an iron fork that jerks down the shoot, and feeds ore into the battery box. A similar appliance is often used for feeding rolls, when a cam driven off the roll shaft takes the place of the striking collar on the stamp stem as shewn in Fig. 120.

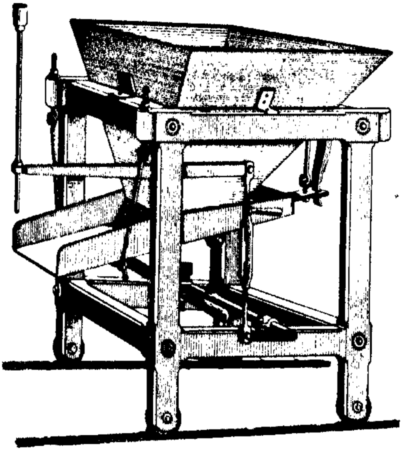
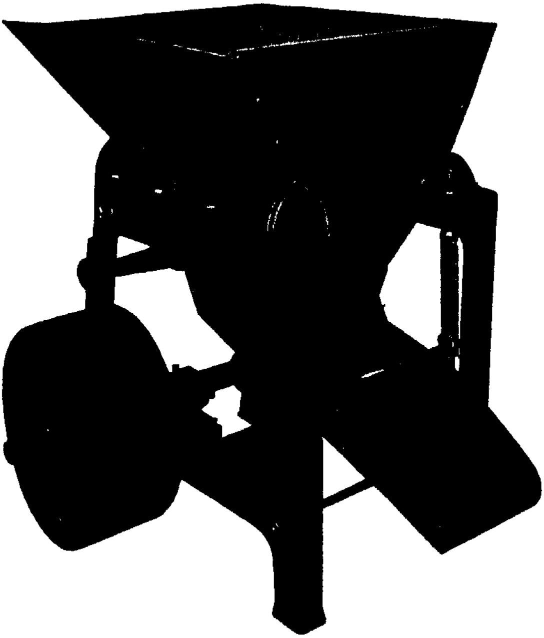


Fig. 379. Tulloch feeder. Perspective.

The **Tulloch feeder**, which was at one time widely used on the Pacific coast of S. America for feeding stamp batteries, and is still used to some extent, shewn in Fig. 379, is of the same type. It consists of a hopper with a swinging tray beneath it, which like the previous one is jerked downwards by the tappet or by a special collar on the middle stamp stem striking the bumping block that projects upwards. When used for rolls, etc., this feeder is built as shewn in Fig. 380; a small driving shaft, worked by a belt off one of the roll shafts, passes beneath the tray.

and keyed on this shaft there is a small cam that throws the tray forward at each revolution of the shaft.

The **Challenge feeder** has already been described on p. 175 in connection with stamp mills, for which it is very largely used, either attached to a hopper or in the so-called suspended form, in which it is applied to the end of a shoot. It may also be used for other forms of machinery, as shewn in Fig. 381, this being a form made by Messrs Fraser and Chalmers, Ltd. A lay shaft is driven by belting, and a cam on this operates the lever actuating the friction ratchet which slowly revolves the oblique table that forms the bottom of the hopper; the revolution of this table feeds the ore forward.



ig. 380 Belt-driven Tulloch feeder. Perspective.

Simpler feeders are made on the same principle in which the table is caused to revolve slowly by means of gearing without the intervention of a cam and friction ratchet. This form does perfectly good work on most ores.

Roller feeders have been made in many forms; they consist essentially of a shoot or hopper closed below by a roller which is slowly rotated so as to carry the material forward; they answer very well on dry, medium sized material, but are apt to feed irregularly with wet sticky ores, and with material very irregular in size. A typical form is shewn in Fig. 382 Sometimes roller feeders are made with rollers having longitudinal ribs or grooves, which seem to feed irregularly sized material more uniformly than do the plain rollers.

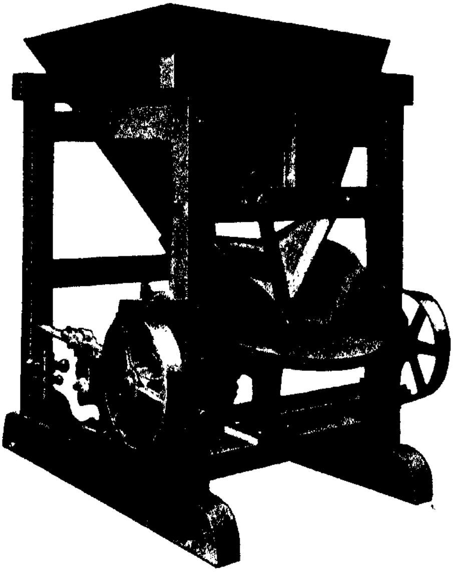


Fig. 381. Belt-driven Challenge feeder. Perspective.

Dryers. It is often required to dry material that is to be submitted to magnetic, pneumatic or electrostatic separation. Drying furnaces are of various types, but two forms, rotary dryers and shelf dryers, are those most generally employed. Rotary dryers are cylinders usually of cast iron, having the portion most exposed to heat often lined with brickwork. The cylinder is usually set at a slight inclination; it is carried on friction rollers and rotated by means of gearing. The upper end is connected to a stack, and the lower end to a fire place. The material to be dried is continuously fed in at the upper end and discharged at the lower, usually into a bin or on to a cooling floor; a very convenient arrangement is to discharge on to an iron conveyor belt, which carries the dried material away and cools it at the same time.

Shelf furnaces consist of a rectangular stack, the upper end being connected with a chimney and the lower with a fireplace. The furnace is filled with shelves which are made in many different forms. The material to be dried is fed in at the top and drops down from shelf to shelf, until it is discharged in the dry state at the bottom. Edison uses a shelf dryer in which the shelves are gently shaken to cause the material to drop readily.

All forms of drying furnace require a good draught so as to carry off rapidly the steam given off during the act of drying.

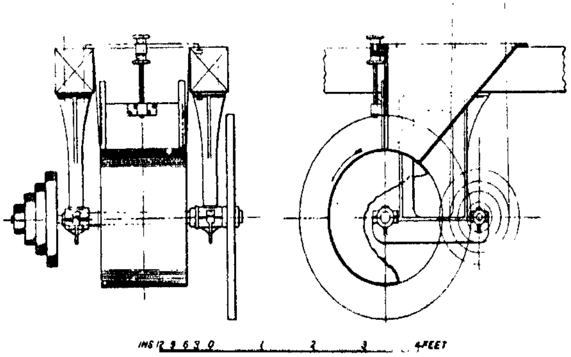


Fig. 382. Roller feeder. Side and end elevations.

Weighing. It is usually necessary to determine the quantity of material delivered to or from a dressing works. This is in most cases best done by measurement, counting the number of cars, buckets, etc., which is usually done by some automatic registering counter, and determining the average dry weight of material that the receptacles contain, such determination being repeated at intervals. When the quantity is determined by weighing the percentage of moisture present must be regularly determined, and the strict rule should be observed in all dressing works to make calculations upon dry weights only. Railway or mine cars are usually weighed on weigh-bridges or platform scales, these being often self-registering. A similar arrangement is used for aerial tramways, a short section of the fixed rail being cut out and hinged.

so that the weight of a bucket hanging from it can be determined. Material that is being carried on a pliable conveyor belt can also be weighed continuously, whilst the belt is travelling, by automatic machinery, a given length of the loaded belt being weighed at each operation.

Sampling. Automatic sampling plant is sometimes employed for taking samples of the crude mineral and of the various products. Such samplers are made on many different principles; the most satisfactory seem to take the form of either a rotating spout that distributes the stream of mineral into a series of pockets or spouts, a hinged spout that is momentarily deflected so as to deliver a certain portion of the stream of mineral into a special receptacle at uniform intervals of time, or else a bucket or tray that cuts at intervals across the stream of mineral. Piles of mineral are usually sampled by hand.

Finally it may be pointed out that a suitably equipped assay laboratory should be looked upon as an indispensable portion of the equipment of every complete dressing works.

CHAPTER XII.

GENERAL CONSTRUCTION OF DRESSING WORKS.

THE conditions under which dressing has to be performed are so various, the minerals to be submitted to it are so different, and the objects to be attained are so widely divergent, that the problem in its entirety presents far too great a degree of complexity to be capable of any general solution. It is rare that the desired result can be reached by any single one of the processes or appliances already described, and the operation of dressing generally consists in a combination of a number of these individual processes. Generally speaking a certain amount of comminution must precede any process of separation, but after the earlier stages, it is not at all unusual to find methods of comminution, methods of sizing, and methods of separation succeed each other alternately. It has already been seen that there are usually a number of different appliances that may be used for attaining a given object, and the choice of any particular one may depend not only on the nature of the materials to be treated, but also upon other conditions, and is frequently influenced by local custom or even by the idiosyncracies of the engineer in charge. Great variations are therefore admissible not only in the processes themselves, but also in their combinations, and to a certain extent also in the order of their succession. Under these conditions it is impossible to lay down any hard and fast rules for the arrangement, design or construction of concentrating works, and it is only possible to offer a few general considerations and suggestions, capable of extended, though by no means universal application.

Site of Dressing Works. It is obvious that the position of such works will be decided mainly by economic considerations; it may be selected with reference to its nearness to the mine or mines producing the crude material to be treated, to the markets or means of communication with the markets, to the water supply, to the ready disposal of waste products, and to the source of motive power. The latter was at one time perhaps the dominant consideration, but with the development of electric power transmission, has become somewhat subordinate to the others. A wet dressing mill must have an ample source of water supply, which is preferably obtained by gravitation, by turning streams or bringing in ditches to deliver to reservoirs above the level of the dressing works; on the other hand when the contour of the country renders this difficult, pumping may be resorted to, especially where power is cheap, and occasionally it may happen that it may be worth while to construct pipe lines of considerable length. Necessarily, since the operation of dressing implies the separation from waste materials, there is at every dressing works such waste to be got rid of, and the ready and economical removal of tailings is a very important matter, which should never be neglected. The omission to provide for it when deciding upon a site frequently entails a heavy addition to working costs after a few years' work. The best situation in this respect is when the works can be placed on the shores of a swamp, a lake or of the sea. A location on the side of a valley down the bottom of which flows a large and rapid river is also a favourable one, provided that the tailings cause no injury to properties further down the course of the river. Sometimes it is necessary to run tailings into dams or settling pits; these should be in duplicate so that one set may be dug out and emptied whilst the others are filling up. This arrangement is sometimes resorted to where water is scarce, the main object then being to use the clarified water over again in the works. Sometimes the waste has to be piled up in big heaps close to the works when no other means of getting rid of it is available, and at times mechanical means must be used for raising the waste material to the top of the heaps.

The accessibility of the site either from the mine or from the means of transport to market is an important matter, the former being the more important consideration when the proportion of worthless to valuable matter is very great (e.g. in a gold ore) and the latter when this proportion is comparatively small, e.g. in a coal mine. It is however usually possible to select a site within reasonable distance of the mine; the mineral is generally brought from the mine either in waggons, by means of hand haulage, horse haulage, self-acting inclines, mechanical haulage, or any other convenient means of transport, or by the buckets of a wire ropeway, the latter method often presenting important advantages. As regards the facility of transport to market, this becomes of vital importance when such products as coal and iron ore have to be shipped. It is almost indispensable in such cases that the dressing works should be so close to a railway line, canal, river, sea, etc., that the dressed mineral can be loaded direct into railway trucks, canal boats, barges or the holds of steamers. On the other hand this consideration is practically unimportant in the case of gold or silver mines.

A few other considerations have also to be taken into account. The site should admit of a good foundation being got, it should be above flood level, and safe from avalanches, whilst the considerations common to all works, i.e. nearness to a source of labour and accessibility to good and cheap supplies, especially fuel, are important.

General Principles of Dressing Works. In the simpler types of dressing works there is scarcely any question of principle involved. There are, however, very many works in which a general succession of operations has to be observed, somewhat as follows:

1. Preliminary coarse breaking of the mineral, which may in many cases be followed by a more or less complete sorting into various qualities.

2. Finer crushing of the mineral.

3. Sizing into sizes suitable for individual treatment, accompanied often by a removal of the slimes for separate treatment.

4. Separation of each size into several—usually three—grades, namely, 1st, more or less clean heads or concentrates, 2nd, middlings for re-treatment, 3rd, waste or tailings, practically barren, or at any rate containing so little valuable material as not to be worth further treatment.

5. Finer crushing of the middlings, followed by further sizing or classification, and separation, until all the middlings are reduced to the form of either concentrates, tailings, or slimes.

6. Slime treatment, which separates the slimes as completely as possible into concentrates and tailings.

The various methods of effecting each operation will be clear from what has preceded; the important subject of middlings already discussed on p. 228 deserves however a little further attention. In the simplest case, where two different minerals have to be separated from each other, many of the machines used to effect this separation are designed to produce practically clean concentrates consisting essentially of one of these minerals, practically clean tailings, consisting essentially of the other mineral, and middlings consisting of a mixture of the two minerals. Middlings may be due to one of two causes: firstly, either the separating machine may be unsuitable to its work or may be given too much work to do, in which case a certain number of grains of each kind of mineral will pass into the middlings, owing to the action of the separating appliance being defective; these middlings, due to some defect in the separating appliance and not to the nature of the material, may be called "false" middlings, and these can be separated into concentrates and tailings by passing them once again through the same machine or through some other machine capable of separating the minerals.

Or secondly, the middlings may consist of grains composed partly of one mineral and partly of the other, and therefore intermediate in properties between the pure minerals, and these "true" middlings cannot be separated into their constituent minerals merely by passing them through separating machines. It is to such true middlings that Prof. Richards' phrase "Once middlings, always middlings" properly applies. What are here called "false" middlings may be, and often are, more difficult to separate than the original crude crushed material, because those particles, which, owing to their size or shape, are the most difficult to separate cleanly, are naturally those that will find their way into the middlings; they are however always capable of separation by suitable appliances. True middlings can however only be treated by crushing them to a smaller size, so as to produce particles consisting wholly of one or wholly of the other mineral. This operation will have to be repeated until the stage of slimes is reached, when the greater portion of the fine particles will consist entirely of one or other of the minerals; the relatively few particles that consist partly of one and partly of the other, will be separated in this sense that particles that consist for the greater part of one kind will pass into the concentrates, whilst those that consist for the greater part of the other kind will pass into the tailings, the dividing line between the two being drawn in accordance with economic rather than strictly technical considerations.

Closely connected with this subject of the re-crushing and further treatment of middlings, is the application of an important principle, known as the principle of "gradual reduction," already referred to on p. 229, the application of which depends upon the character of the mineral to be treated. If the mineral mass consists of very minute particles of the individual minerals uniformly disseminated, it is obvious that the whole mass must be crushed down to at least the size of these particles to effect any separation. But if the particles of one of the constituent minerals, even though small, be irregularly distributed, it is obvious that larger fragments can be found, which would consist practically, if not absolutely, of one or other of the minerals in question; generally speaking the more valuable mineral forms the smaller proportion of the mass, so that comparatively large fragments of comparatively clean waste may mostly be obtained. It would therefore be possible to first of all break the mineral to a suitable, comparatively coarse, size, and to eliminate at this stage a certain proportion of practically clean waste; the remainder would then be crushed smaller and the process repeated, a certain amount of waste being got rid of at each stage, until the remaining particles are small enough to enable a final separation to be made. It is obvious that by this process of gradual reduction, only a small proportion of the whole mass need be crushed to the degree of fineness necessary for complete separation; of course a certain proportion of the valuable mineral is liable to be lost with the waste, but the pecuniary value of this may well be less than the additional cost that would be incurred by fine-crushing the whole mass, so that the adoption of this principle would prove to be directly profitable. Furthermore the capacity of a given plant is also increased thereby, and there is less fear of valuable material being lost in the slimes. The principle of gradual reduction may be applied in another way, in dealing with a mineral mass in which the valuable ingredient is in part disseminated in coarse particles. Relatively large fragments of the comparatively clean valuable mineral may be obtained in the earlier stages, whilst the remainder is entirely contained in middlings that need further crushing. This presents the advantages that a good deal of the valuable mineral is obtained in a coarse state in which it may fetch a higher price, whilst there is less risk of loss through sliming a part of it.

Again it may happen that both these modifications of the same principle may be applied; the comparatively coarse fragments may be separated, yielding a portion both of the mineral it is desired to save, and of the waste in a comparatively clean state, that is to say as finished products, whilst the middlings alone will need crushing down smaller.

The methods for the treatment of slimes vary even more than those for the separation of the coarser particles; the principles of slime treatment are still imperfectly understood, and in each case the best method has usually to be got at by experiment. Slimes are usually submitted in the first instance to classification, and each class is treated by itself on a suitable appliance, the action of which obviously depends on some other principle than that of equal-falling particles; very often some form of shaking table is used for the coarser grades, and some table with a travelling surface (vanners, Harz tables, etc.) for the finer grades. The finest slimes, too fine to be collected in any ordinary classifier, may still carry too much value to be treated as waste. In such cases, unless special methods can be applied, these very fine slimes may be allowed to settle slowly in labyrinths, slime pits, reservoirs, etc., and the mass dug out from time to time and dressed on a suitable machine; it is clear that this process should only be carried on when the value of the mineral saved exceeds the cost of collecting it.

The principles of dressing have here been discussed as though the problem were always the separation of two minerals from each other; in practice it often happens that three or four (rarely more) minerals have to be separated from each other. In such cases the basal principles are necessarily the same, but the operations assume a form of greater complexity, there being often middlings of two or three different kinds produced, each of which will ultimately contain two different minerals only, the separation of which proceeds upon the lines already indicated. For example, in treating a mineral containing galena, zinc blende and quartz, the products of a first dressing would be somewhat as follows: practically clean galena, No. 1 middlings containing galena and zinc blende, practically clean zinc blende, No. 2 middlings containing zinc blende and quartz, and tailings of practically clean quartz. Even assuming that the galena, zinc blende and tailings are clean enough, the two former for the market and the latter for running to waste, without any further treatment, the two classes of middlings will have to be submitted to further crushing and separation, but in practice it will be found that every one of the first four products, and often the tailings also, will need further treatment.

Design of Dressing Works. This consideration again affects only the more elaborate forms of works, the design of the simpler forms needing but little attention. All that is required in such is to so lay out the works that the minimum of labour shall be required in dealing with the mineral. Thus the road on which the cars containing the mineral enter the dressing works may be laid at such a gradient that they can run down it automatically and thus pass to a tippler without any attention. The tippler is best of the revolving type, and is often so arranged as to be automatically thrown into gear by the motion of the cars themselves, and in some cases after its contents have been tipped the car is automatically run forward till it reaches some form of elevator—often a creeper—by which it is brought back to the original level at which the cars enter. The tipped mineral may pass automatically to the breaker, sorting or washing appliances, and thus the whole work be done with the minimum of labour.

In the more elaborate works the design both in plan and elevation has to be considered. In large dressing works it is best to arrange the works so as to be duplex, triplex, etc. in plan, i.e. to lay it out as a number of units, each of which is practically complete in itself, and can be stopped or started independently of the other units. The duplex arrangement is a favourite one, the works being arranged in two halves each of which is a counterpart of the other. For example, there may be a central breaking department, or breaking and sorting department; the product from this may fall into a large main bin, from which it may go by two spouts, one to the right and the other to the left, to symmetrically disposed crushing, sizing and separating machinery. The slimes from these two sides may be treated either in two separate slime dressing plants, each corresponding to either side of the main works, or the slimes from both sides may be united in one common receptacle, and go either to one slime-dressing unit, or to two symmetrically arranged ones, according to the quantity to be dealt with.

Two different schemes of design in elevation are in vogue; the one adopts a series of terraces on a suitable hill side, the other erects the mill on a level site. The advantage of the former method is that the bulk of the mineral to be treated passes by gravitation from each appliance to the succeeding one. Thus the upper floor or terrace may be taken up with bins for containing the crude mineral; from these it passes to the breaker floor, whence the broken material falls to the next level, which forms the sorting floor; the next lower floor is the crushing floor, the floors next below are taken up with sizing and separation, whilst the lowest floors are devoted to slime treatment, and the finished products are shipped at a level below this. Generally speaking a slope of 1 in 3 is needed for such a series of floors or terraces; these have to be excavated out of the hill side, or else formed by means of retaining walls, or both methods may be combined. The former is usually to be preferred, as it is more likely to admit of substantial foundations being obtained, this being a point of the utmost importance. Not only are solid foundations required for carrying heavy breaking and crushing machinery, notably in the case of stamp mills, but there are also very few dressing appliances that will do satisfactory work unless they are erected on foundations so substantial as to prevent any shake or vibration being set up in the framework of the machines. The advantage of all the material passing continuously downwards in its progress through the works is very obvious, especially where wet methods of dressing are employed. It must, however, not be forgotten that it is almost inevitable that certain portions of the material must be sent back for further treatment, e.g. over-sizes for further crushing, middlings for finer crushing and further separation, partially dressed products for final treatment, etc. All such material must necessarily be elevated for further treatment, and where the proportion is at all large it may require so much lifting as to neutralise the advantages of the gravitational arrangement.

The other scheme of design consists in selecting as flat a site as possible, all the works being built on one level. It is, of course, possible to build a works on a level site having several floors, and to elevate the mineral once for all to the top and then to allow it to descend under the action of gravity. This method combines, however, most of the disadvantages of both systems, with the further drawback that it is practically impossible to build the upper floors stiff enough to form really satisfactory foundations for the heavy machinery that they have to carry. It is better, where the flat site is adopted, to design the works on an appropriate principle, and to build them practically all on one level, only those products that need lifting up at each stage being elevated by suitable means. The different operations are then carried on in a number of buildings or rooms adjoining each other, so arranged that the crude material enters at one end and issues in the finished state at the other, there being no travelling to and fro except for such portions as require re-treatment. This system has many advocates; it usually allows of good foundations being obtained, provides a structure readily accessible from all sides and gives a greater latitude for the selection of a site than when a hill side of definite gradient has to be sought for. It occupies usually a good deal of space, but as dressing works are generally erected in districts where ground is practically valueless, this is no real drawback, whilst if properly laid out the mineral need travel no further in its way through the works than by the former plan. It often enables works to be built on a site more convenient for communication with means of transport than the first method. On the other hand it may entail a great deal of elevating machinery, which is usually troublesome to keep in repair, and costs a great deal both for power and for wear and tear. It further involves often the pumping up of considerable quantities of water. The great drawback to it is, however, the difficulty that it often presents to the disposal of tailings and waste rock.

When the system of building in floors on a flat site has to be adopted, provision should always be made for the first breakings, requiring as a rule heavy machinery, to be performed on the ground level, the broken material being subsequently elevated. This plan presents the further advantage that broken mineral is more easily elevated than the same material in coarse lumps. Neglect of this consideration entails in most cases either defective breaking plant due to unsatisfactory foundations, or else great expense in the form of lofty massive piers or walls to carry this machinery.

In this connection the site of the breaking machinery deserves attention. In important undertakings it is becoming the modern practice to separate entirely the coarse breaking plant from the rest of the dressing works. This method is well illustrated in the large gold mills of the Witwatersrand district, where the breaking and sorting is now usually carried on in crusher stations, often at a considerable distance from the mills proper. Such crusher stations may be situated at the shaft top and may form almost a portion of the shaft equipment, or they may be situated at a point convenient of access from two or more hoisting shafts; the mineral is there broken and sorted, and only that portion intended for further treatment is transported to the mill. This system presents many advantages, especially on flat sites and where electric transmission of power is employed. The breaking operation is a rough one, inseparable from much shock and jar and the production of a great deal of dust, all of which are highly injurious to the more delicate operations of the dressing works proper. Even where a remote crusher station is not indicated, it is always advisable to perform the breaking in a building distinct from the dressing works. Where sorting is also needed a very good plan, now much in vogue, is to connect the breaking plant and the dressing works by a long conveyor belt, upon which the picking is performed, and which keeps the two departments sufficiently apart.

Construction of Dressing Works. The importance of good foundations has already been referred to, and it is always best to secure these when possible by digging down to solid rock and laying down the foundations—practically always now of concrete—upon this. When a rock bottom is not available, very massive concrete blocks should be substituted, expense incurred in this way being money well laid out.

The buildings themselves are of very varied type, practically all forms of structural material having been employed. In countries where wood is cheap (e.g. United States and Scandinavia) the buildings are usually wholly of wood, substantially framed, the sides being either of vertical boards, grooved and tongued, or clapboarded, more rarely shingled; the roofs are nearly always shingled. This is a very suitable construction, except for the great liability to fire; wooden buildings can readily be made weather-proof and easily kept warm in cold climates. Another very usual form is a wooden frame with sides and roofs of galvanised iron, whilst for large mills a framework of structural steel covered with galvanised iron is often used. These buildings are very difficult to keep warm in cold weather; the latter offer a fair measure of security against fire. Brick buildings, or occasionally stone, with tiled roofs, are often used for smaller works, and are quite satisfactory, but are usually considered too expensive for large dressing works. It may be surmised that in the future important works will be constructed in ferro-concrete, a material that is admirably adapted to the purpose. The roof must not rest upon any portion of the framework of the machinery; it is best not built all in one plane, but broken up into several bays, especially where heavy wind storms are liable to occur. This arrangement also affords good opportunities for ventilation and for getting a top-light. Illumination both by day and by night should be carefully studied. A top-light from Louvre windows is the best, but there should also be numerous large windows in the sides and ends, and the machinery should be planned so as to interfere as little as possible with the light from these. For night work electric lighting is far superior to any other, and is generally employed. Clusters of from 3 to 5 powerful incandescent lamps (30 to 100 C.P.), or a number of flaming arc lamps, are the most satisfactory and far preferable to using only a few powerful arc lights, as it is important to get the light as well distributed as possible. The floors should be watertight, preferably of concrete; where wood has to be employed the seams should be caulked like a ship's deck. They are best set on a slight slope towards a gutter, so that any leakages or splashings of valuable mineral can be recovered without difficulty.

Many dressing works are situated at high altitudes, so that some provision for warming them is generally required; this is often done merely by stoves or braziers, but in a well-made building steam heating may be employed with advantage, especially in wooden buildings where every possible precaution should be taken against fire.

Wet dressing works necessarily require an abundant supply of water; the quantity varies within a very wide range, wet dressing requiring from less than 2 to more than 20 times as much weight of water as of mineral treated. Such water should be as pure and as clean as possible, and if muddy should be allowed to clear itself in settling ponds. Sea water may be used in case of need instead of fresh water; it has been used quite satisfactorily in various cases. The water supply should be delivered to the works from tanks or reservoirs situated at a good height above the top of the works, a head of 20 to 50 feet being satisfactory. Even in dressing works that do not use wet methods a good supply of water is desirable both for washing down the floors and for use in case of an outbreak of fire. It is best to have on every floor of the works two or three standpipes with hose attached, so that a stream of water can be at once brought to bear on any part of the works.

Motive power. In practice power is supplied in three ways, namely, by means of water, steam, or electricity, according to circumstances.

It is a moot point whether it is better to drive the whole of the dressing works by means of suitable belting from one main motor, or whether it is better to have a number of motors commanding the principal lines of counter-shafting. Where electricity is employed there is little doubt that the latter is the better plan, each main line of counter shafting being driven by an independent motor. The cost and complexity of a quantity of belting are thereby avoided, each shaft can be stopped, started or run at any desired speed, independently of the rest of the mill, less power is wasted in belting and counter-shafts, whilst electric motors of moderate size can be made almost as efficient as large ones, and no more cost is incurred for attendance as the motors need practically no attention, except for cleaning and lubrication. In the case of steam and water, however, the case is different. It is not easy to instal a number of separate small hydraulic motors, and their efficiency would be less than that of one or two larger ones. Where steam power is used, it may be laid down that in any case all the boilers required should be concentrated in one boiler house, so that the employment of a number of separate steam engines would necessitate carrying steam from the boiler house to each engine. Loss of steam by condensation is almost inevitable, and small engines are generally less efficient than large ones. In these cases it is therefore generally preferable to have but few motors and to transmit the power to the various line shafts by means of belting. In plants using rock-breakers (and the same is true to a lesser degree of all crushing machinery) it is very important that the power driving these should be obtained from a motor that does not at the same time drive the dressing machinery proper. Such breaking operations necessarily make very varying demands upon the power supply, requiring, for example, a great deal of power when a quantity of hard mineral in large lumps has to be broken, and very little power when only soft material in small pieces is being put through. This causes rapid, sudden and irregular fluctuations in the speed of the motors, which do but little harm as far as breaking is concerned, but which would seriously interfere with the efficiency of any dressing appliances, which have to be kept running steadily at quite uniform speeds. Separate motors for the crushing machinery and the dressing machinery proper should therefore always be provided, even when all are together in one mill. The motors driving the dressing appliances should be provided with sensitive governors so as to keep the speed uniform.

Power is best transmitted from the main motors by rope drive; where belting is used, canvas (Gandy), camel-hair, or indiarubber are preferable to leather in places where they are liable to get wet.

Where water power is obtainable this is undoubtedly the most satisfactory. The older dressing works in England used over-shot water wheels almost exclusively, and these give very satisfactory results when a great amount of power is not required and when considerable quantities of water with comparatively low falls are available. The efficiency of a good over-shot wheel may be taken as 65 per cent. Many dressing works are situated in mountainous districts where very high falls are to be obtained, and these are best utilised by impact wheels of the Pelton type, these being cheap, simple, economical to run, and highly efficient; they may be applied to all falls of more than 50 feet in vertical height, though they are most efficient for falls of 100 feet and upwards. In practice under such circumstances an efficiency of 75 per cent. may be reckoned upon. For lower falls turbines are best employed; a twin Jonval parallel-flow turbine on a horizontal shaft offers many advantages, amongst which may be reckoned the fact that the turbine itself may be situated 10 to 15 feet above the level of the tailwater and therefore often above the reach of an ordinary flood, without any loss of power, by the use of a properly arranged suction tube. For very low falls, say 5 ft. or thereabouts, radial-flow turbines on vertical shafts are often employed. The efficiency of turbines varies according to the type employed and the conditions of installation, but may be averaged at about 70 per cent. Under suitable conditions water power is extremely cheap, costing from £1 per horse-power year upwards; the great drawback to it is that it often varies very greatly at different seasons of the year.

Steam power is employed in districts where water power is not available and where fuel is reasonably cheap. In large modern works water-tube boilers with automatic stokers, superheaters and feed-water heaters are employed, and high pressure steam is generated which is utilised in compound or triple-expansion engines which are often condensing. For smaller installations a less efficient plant has necessarily to be employed. The boiler house should always be a substantial building, separate from the dressing works so as to diminish the risk of fire. The cost of steam power varies within very wide limits dependent upon the price of fuel and the nature of the plant. A large compound engine will generate 1 I.H.P. per hour with 12·5 lbs. of steam, and good boilers with fair coal will evaporate 10 lbs. of water for each 1 lb. of coal burnt, so that 1·25 lbs. of coal will be consumed for each I.H.P. hour. With coal at 15s. per ton the cost will be about £6 per horse-power year for a 24 hour day in large plants under favourable circumstances.

Electricity has been growing rapidly in favour as a motive power during the last 15 years; owing to the ease and economy with which high tension electric currents can be transmitted, dressing works can now be located at any desired point irrespective of where the power is actually generated. The generating station may be driven by water power, where this is available, by steam, in which latter case modern practice favours turbo-generators, or by gas engines worked by producer gas, the latter offering the great advantages of low fuel consumption and of being able to use fuel of inferior quality to greater advantage than can steam boilers.

It is probable that in the near future suction gas producer plants may be employed with advantage for driving small dressing works.

The quantity of power required varies within very wide limits; as the greater portion is required for comminution, it depends greatly upon the degree of fineness to which the mineral has to be crushed. Thus in a works like a stamp mill for gold quartz, in which the whole of the ore has to be crushed very fine, the efficiency is from 1 to 1½ tons per 24 hours per I.H.P.; in a dressing works using crushing rolls and jigs the efficiency is from 3 to 5 tons per 24 hours per I.H.P.

The above principles may be illustrated by brief descriptions of dressing plants used in the treatment of the more important minerals.

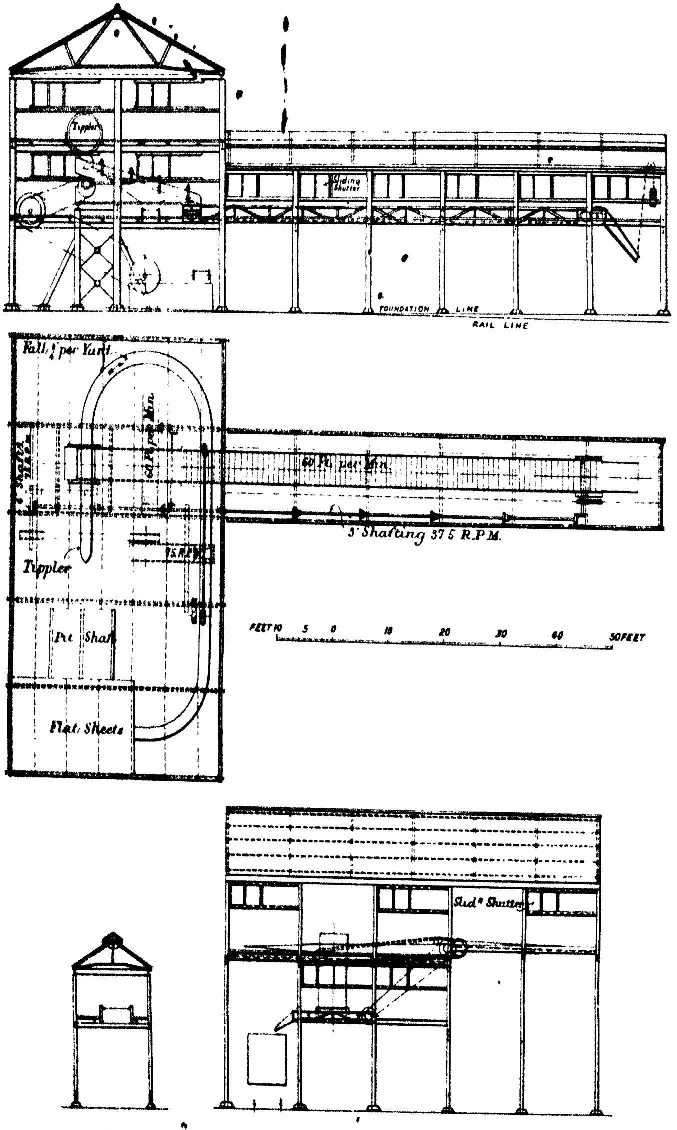


Fig. 383. Colliery Heapstead. Side and end elevations and plan.

CLASS I. FUELS.

Coal. A great deal of the coal mined merely requires screening so as to make one or more sizes as may be required by the market, whilst any shale, waste rock, fireclay, lumps of pyrites, inferior coal, etc. has to be picked out. A small simple modern "heapstead" for this purpose is shewn in Fig. 383, which represents a design by Messrs Joseph Cook, Sons and Co., Ltd., in plan, side and end elevations. The tubs as they come from the shaft gravitate down through a revolving tippler, and continuing to run down the curved track, run automatically to the foot of a creeper (see p. 463), which brings them up again to the level of the flat sheets on the opposite side of the shaft. The coal from the tubs is discharged by the tippler spout on to a plain jigging screen. The undersize from this is taken away by a narrow cross belt, whilst the oversize drops on to a picking belt at the end of which a movable shoot discharges it into railway waggons, the plant being capable of dealing with 500 tons per day.

A similar but larger heapstead by Messrs Coulson and Co., Ltd. is shewn in plan and various elevations in Fig. 384. As shewn by the plan, the colliery tubs brought up the shaft travel by the two roads shewn (one for each cage) in the direction of the arrows, these roads being laid at such a gradient from the shaft that the tubs run down automatically; they thus run on to the foot of the creeper, by which they are elevated sufficiently to enable them to complete the round journey back to the shaft without needing the application of any force other than gravity. They run from the top of the creeper to a buffer, from which they return by means of an automatic switch along the road leading to the tipplers. There are three different qualities of coal drawn from this pit, and the large coal of each class has to be kept separate. A lad switches the tubs as they come along on to their proper road, there being a tippler for each class. When the tubs leave the tippler they continue their journey by the "empty" roads as shewn, and thus return to the opposite side of shaft to that from which they started. The coal from each tippler falls on to a jigging screen, the oversize from which goes to its proper picking belt, which is provided with a movable jib end for loading it direct into railway waggons. The undersize from the three screens falls on to a transverse conveying belt, that from the middle screen, which is set further back than the two others, being delivered to the same belt by a vibrating shoot. The conveying belt

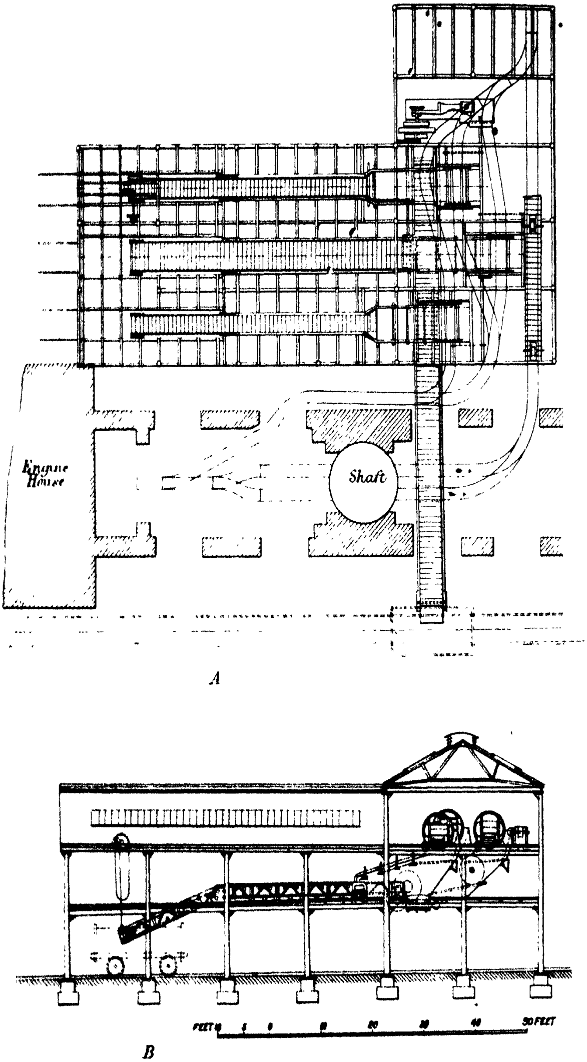
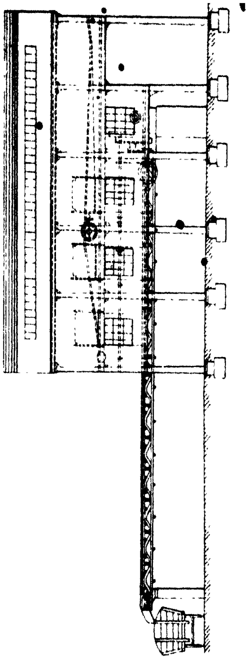


Fig. 384. Colliery Heapstead. *A*, Plan. *B*, Side elevation.

carries the small coal direct to a shoot whence it drops into railway waggons. This heapstead is capable of dealing with 1000 tons in 10 hours.

It will be noted that these heapsteads are laid out so as to work with a minimum of labour, and this is typical of the best modern practice. Sometimes the small coal from each screen has to be kept separate, and sometimes several sizes, such as lumps, nuts, peas, duff, etc. have to be made, and kept separate. In these cases a correspondingly larger number of transverse conveying belts, each delivering on to a separate railway track, will have to be employed. Too great a degree of complication is, however, usually to be deprecated. It is often considered better to lay the heapstead out in such a way, if possible, that the full tubs run automatically to the tipplers, and that only the empty tubs have to be "crept" up to the banking-out level, as it takes less power to elevate empty tubs than full ones.

A heapstead on the above principle is quite sufficient where only large coal, say over 1½ inch cube, has to be



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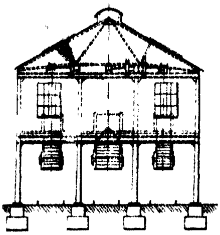

D

C, Elevation of tippler end shewing cross-belt.

D, Elevation of discharge jib-ends of main belts.

dealt with, but when small coal has to be cleaned, e.g. especially for the purpose of coking or briquetting, some washing arrangement has to be adopted. In the simpler and cruder forms the small coal is washed just as it comes through the screens without further sizing, whilst in the more elaborate, but also more costly plants, the coal is sized before it is washed, thus cleaning the coal more effectively.

A very simple arrangement is shewn in Fig. 385[1]. The coal tubs are tippled on to a jigging screen, 14 feet by 8 feet, with a gradient of 1 in 5, making 90 four-and-a-half inch strokes per minute, the bars being ½ inch apart. The oversize (about 250 tons per day of 10 hours) goes to a circular picking table, 10 feet in outside diameter and 20 inches wide, making three revolutions per minute; there are six pickers at this table. Any pieces of mixed coal and dirt are thrown through a shoot in the middle of the table to a lower platform where they are broken and picked by three pickers, who also fill the dirt into tubs. The undersize from the jigging screen (about 100 tons per day of 10 hours) drops into a trough where a stream of water meets it and washes it down to the simple trough washer (shewn in detail in Fig. 237, p. 302) with two compartments, used alternately, each being 17 inches wide, 13 inches deep and 150 feet long, set at a gradient of 1 inch to the yard. The water consumption is about 400 gallons per minute, and the velocity of flow about 300 feet per minute. The washed coal runs into a box from which it is lifted by a bucket elevator worked by a 6-foot water-wheel driven by the overflow of the washing water. A centrifugal pump pumps the water back. About 18 tons of dirt are washed out per day, and the washer is worked by one man and two boys.

Fig. 386 is another example of a simple plant, being a two-trough Elliott coal washer, built by the Hardy Patent Pick Company, Ltd. The coal to be washed is delivered to a hopper from which an elevator lifts it to a couple of bins, placed one over each washing trough; the dirt and shale are discharged at the upper end of the troughs into a shoot through which it drops into a truck. The clean coal at the bottom end drops on to a shaking screen with fine mesh, through which the water is drained off, whilst the coal drops into a boot, from which it is elevated to storage bins. The water drained off from the washed coal flows to a settling tank, where any fine sludge can settle, whilst the water is pumped up by a centrifugal pump to be used over again. The capacity of this plant is about 150 to 200 tons of small coal in 10 hours.

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[1] *The Mining Inst. of Scotland*, Vol. XI. p. 182.

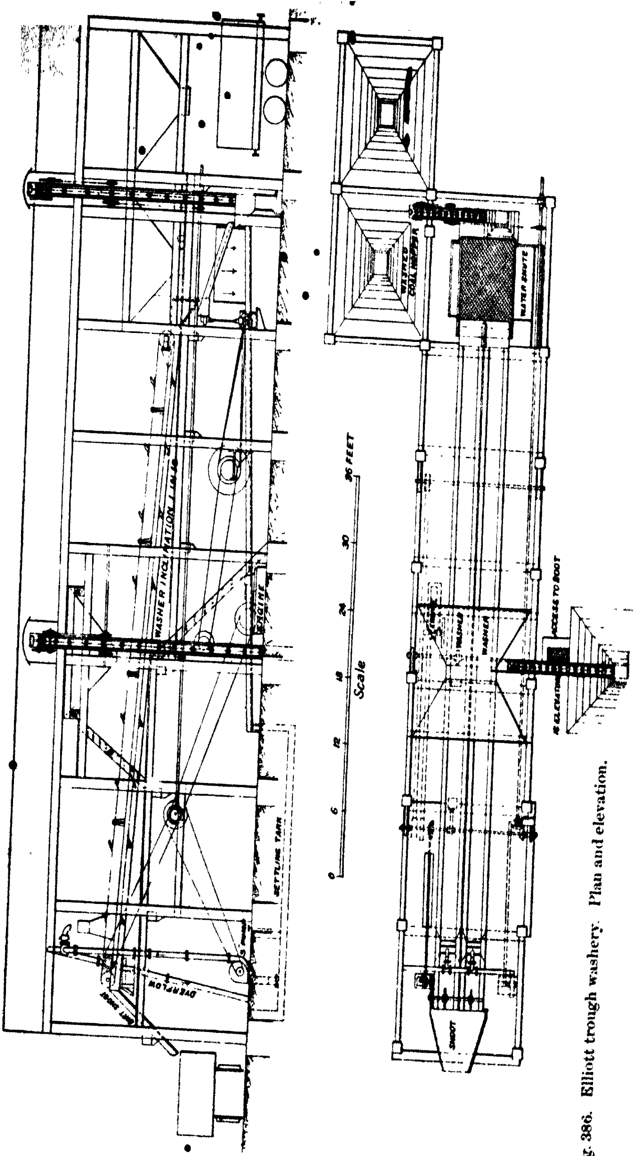

Fig. 386. Elliott trough washery. Plan and elevation.

It is obvious that a similar general arrangement can be employed with any other form of trough washer.

The Robinson washer (see page 246) is often used instead of a trough washer for this class of work, the general arrangement of the plant being somewhat similar, as may be seen from Fig. 196.

Such a plant will wash about 20 tons per hour, the ash in the unwashed coal being from 10 to 14 per cent., and in the washed coal from 4·6 to 5·7 per cent. One man can attend to such a plant, exclusive of the labour of bringing in the coal and removing the coal and dirt. The first cost of such a plant is about £300.

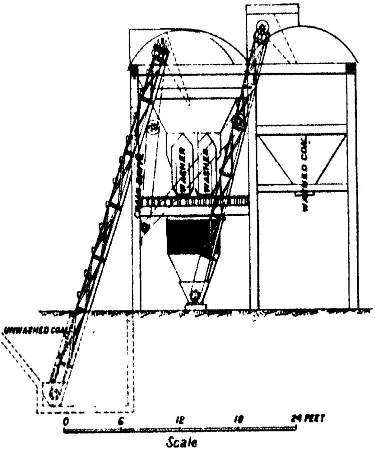

Fig. 386^{a}. Elliott trough washery. End elevation.

As an example of the more complicated form of washery, a plant built by Messrs Sheppard and Sons for a South Wales Colliery may be instanced. The coal is screened at the colliery heapstead, the small coal, that has passed through 1¼ inch screens, being brought in railway waggons to the washery, where it is tipped into the storage bin *A*, Figs. 387 to 389. It is lifted by a bucket elevator *B* into the narrow end of the compound conical trommel *C*, 16 feet long, which is fitted with ⅜ inch and ¾ inch wire mesh screens. There are thus three sizes made: nuts, ¾ inch to 1¼ inch, 20 per cent. of the whole; peas, ⅜ inch to ¾ inch, 20 per cent.; and duff, below ⅜ inch, 60 per cent. The two coarser sizes are washed each in a two-compartment nut washer *J*, running at 60 to 70 five-inch to seven-inch strokes per minute. The dirt is raised by an elevator *R* into a hopper, whence it can be discharged into waggons, whilst the coal passes to a trommel *L* in which the water is drained off from it, and the coal is then transported by the screw conveyor *M* into the storage bin *N*. The duff, or undersize from the trommel *C*, passes by a long screw conveyor *D*, to ten double felspar washers *E*, working at 180 to

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190 quarter-inch strokes per minute. The dirt drops into troughs common to each set of hutches of the washers and is carried by a screw conveyor to four elevators *F*, with perforated buckets, which deposit it in the same bin that receives the dirt from the nut washers. The coal is carried by the stream of water to a large settling pit *G*, 45 feet long, with a depression at one end in which works the elevator *H* which lifts the washed coal up the screw conveyor *O*. A scraper works in the bottom of the settling pit carrying all the coal deposited there to the boot of the elevator. A centrifugal pump *I* pumps the clarified water from the settling pit back again to the top of the building. The plant has been found capable of washing 450 tons of coal per day of 10 hours, extracting 9·6 per cent. of refuse therefrom.

A still more elaborate washery is shewn in Figs. 390 to 392[1], representing one built by the Lührig Coal-Washing and Ore-Dressing Appliances Co., Ltd. The coal is tipped from the colliery tub in tipplers from a platform *a*, and is screened on three 2 inch jigging screens, *b* the oversize passing on to three picking belts of the ordinary type, and the undersize dropping into a large storage hopper *c*. The inferior coal picked out on the belts is broken on a breaker *d*, and this and the small coal are elevated by the bucket elevator *e* to a complex trommel making nuts down to about 1½ inch mesh, bears down to about 1 inch and peas down to about ⅜ inch mesh, each of which is washed in a separate felspar washer *f*; the clean coal is delivered over drainers *g* into hoppers *h*, whilst the middlings, consisting of dirty coal, are conveyed by a screw conveyor to an elevator that lifts them to the crushing rolls *i*; the crushed material is washed on a separate nut washer. All the waste is lifted by the elevator *k* and discharged. The fine coal, the undersize from the ⅜ inch screen, is classified in a spitzkasten *m*, and a series of spigots supply the fine coal washers *n*; the waste is removed by the elevator *o*. The fine coal is carried by the escaping water to a drainage drum, which separates out the coarser portion; this is then lifted by an elevator *p* into storage hoppers *q*. The finest coal settles in a big settling tank *r* with a depression in the end, forming a boot for the elevator *s* which lifts it to the storage hopper. From the other end of this settling pit clear water overflows into a tank and is pumped up by the centrifugal pump *t*. The engine-house containing the lighting dynamo is shewn at *w*. The plant (inclusive of the picking belts) treats 1500 tons per day of 10 hours.

It will be noticed that the main difference in principle between this plant and the previous one is that the dirt from the nut washers is treated as middlings and is crushed and re-washed, so as to save the coal that it contains.

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[1] *Engineering*, Feb. 13, 1891, p. 184.

Another plant on similar principles, built by the Humboldt Engineering Works Co. for a French colliery is shewn in Figs. 393 to 396; it is intended to treat in 10 hours 700 tons of small coal which has passed through a 40 mm. (1·575 inch) screen. The small coal is dumped from the waggons into a storage hopper, whence a conveyor takes it to the boot of a bucket elevator. This delivers to a complex trommel, which makes five sizes, namely 1·5 to 1·2 inch, 1·2 to 0·79 inch, 0·79 to 0·47 inch, 0·47 to 0·24 inch, and the undersize from 0·24 inch. The four coarser sizes go to four nut washers, and the clean coal from these goes into fixed draining screens which deliver it to a series of pockets. The finest size is washed on fine coal jigs and the washed coal from these is run into a series of bins in which it is drained, there being a number of pockets in which the fine coal is stored until sufficient water has drained off it. The fine coal is then run out from the bins into a scraper conveyor that takes it to a special fine coal elevator. The fine tailings are taken direct to the dump, the fine middlings and the coarser middlings and tailings being drained, crushed and washed in special washers. Arrangements are provided by which each size can be loaded separately or all sizes together as may be required. The water runs into a series of settlers of the spitzkasten type, where the slimes settle out, the clear water being pumped back again to the washery.

Another very similar plant, built by the same firm for a South Wales colliery, treats all coal below 1¼ inch mesh, and is capable of treating 500 tons of coal per 10 hour day and is driven by a 160 H.P. tandem compound engine.

Coppé's washeries are arranged on practically the same principles as the Sheppard, Lührig or Humboldt plants. Baum's washeries differ in that the coal is washed first and then classified, the fine coal under ½ inch being then washed again.

Fine washed coal is suitable either for coking or for making into briquettes, forming what is often spoken of as "Patent Fuel." Briquetting is usually performed by mixing the fine coal with a small proportion of pitch (usually about 6 per cent.) and heating the mixture which is then exposed to great pressure in a suitable press, forming blocks of various shapes and sizes.

Anthracite requires a treatment differing from bituminous coal; it burns less readily, and requires a large air supply, which can be obtained by burning it in comparatively small pieces, of very uniform size, so as

to expose the greatest possible surface of coal for a given mass. Hence anthracite has always to be broken, and very closely sized. It is usually broken in rolls, fluted or toothed, and sized on shaking screens, Coxe screens, or some similar form of screening appliances; trommels are said to wear out very rapidly. In America the sizes made are generally about as follows: lump, over 5 inches; steamer, 5 inches to 4 inches; broken, 4 inches to 3½ inches; egg, 3½ inches to 2 inches; stove, 2 inches to 1⅛ inches; chestnut, 1⅛ inches to ¾ inch; pea, ¾ inch to ½ inch; buckwheat, ½ inch to ¼ inch; rice, ¼ inch to 3/16 inch; barley, under 3/16 inch. The sizes are only approximate, and vary a good deal, while special sizes are at times also made. The larger sizes are picked by hand, or the slate is separated by mechanical means, depending on the fact that the accompanying slate is in thin flat pieces, whilst the pieces of coal are more nearly cubical in shape. The medium and small sizes are cleaned by jigging like bituminous coals.

CLASS II. ORES.

Iron Ores. Many of these need no treatment at all, as for example in some of the mines in Michigan, U.S.A., where the ore is loaded by steam shovel, direct from the mines into railroad cars. In other cases, e.g. in the Cleveland district of Yorkshire, hand picking alone suffices, the ironstone being tippled direct on to a picking belt or similar device.

Some ores, particularly brown haematites, occur in deposits in a tough adherent clay that has to be washed off, for which purpose any of the washers described on pages 100 to 106 may be employed, the arrangement of the plant being usually extremely simple. Such ores are met with, for example, in Virginia, U.S.A., and in the province of Santander, Spain.

A log-washing plant as used in Longdale, Virginia, already referred to on p. 101 is shewn in Figs. 397, 398[1]. The ore is brought in railway cars on to the trestle *T*, Fig. 398, and dumped into the hoppers *U*, *U′*, from which it passes to the lower ends of four log-washers. The logs are iron pipes 17 feet 5½ inches long, 11½ inches in diameter, of ¾ inch metal, as shewn in Fig. 397, the paddles being set in a double spiral with a 5 foot pitch; they make 12 revolutions per minute. The troughs, *R*, are made of semicircular cast iron plates, bolted to wooden frames. At the top end of the washers the ore is discharged into four drainage trommels *Q*, of 3/16 inch steel plate, punched with 3/16 inch holes. The ore drops from the end of this into a shoot delivering to the ore cars. The undersize drops on to a 20 mesh screen at W, the oversize from which also goes to the ore cars, whilst the water runs to waste through the trough Y. The muddy water running off at the lower end of the trough is run over a screen to collect any fine ore that it may contain, and then also runs to waste. The whole plant is driven by a 25 H.P. engine and needs the labour of six men. It washes about 200 tons of ore per day, the production of washed ore being from 70 to 75 per cent. The coal consumption for steam raising amounts to 10·6 lbs. per ton of ore treated.

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[1] *Trans. Amer. Inst. M. E.* Vol. XXIV. 1894, p. 34.

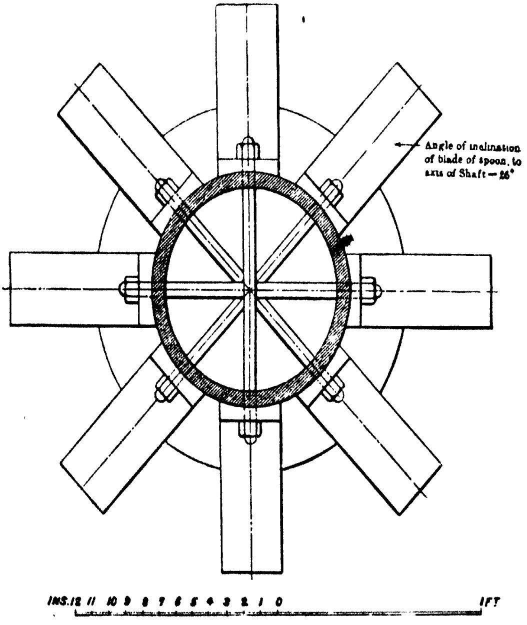
Angle of inclination of blade of spoon, to axis of Shaft = 26°
INS. 12 11 10 9 8 7 6 5 4 3 2 1 0
1 FT

Fig. 397. Longdale log washer. Section of log.

A large plant at Santander uses drum washers, as mentioned on p. 105. The ore-clay is brought in railway cars on to a trestle, whence it is tipped on to a platform, off which it is sluiced into eight large washing drums, 13 feet long by 6 feet 6 inches in diameter, with a conical discharge 5 feet long, tapering down to 1 foot 8 inches. The ore-clay is sluiced into the wide end of these drums over a grizzley with bars 4 inches apart, any large lumps of ore remaining on these being picked out by hand. The eight drums treat 3000 to 3500 tons per day, the water consumption being 500 tons per

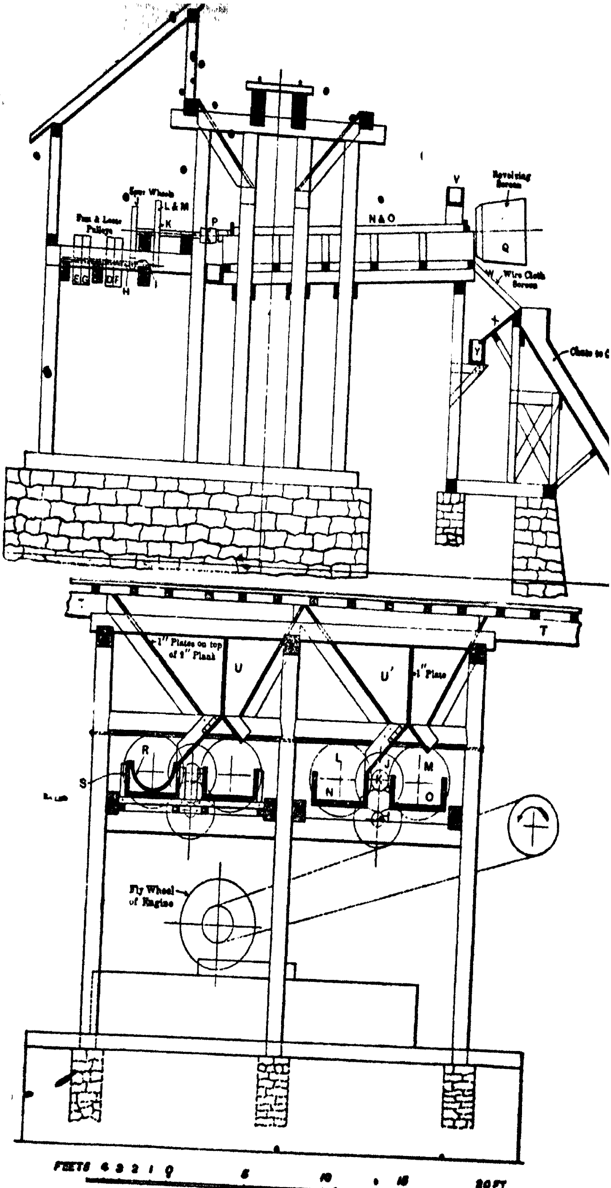

Fig. 398. Longdale log washer. Longitudinal and transverse elevations.

hour. The plant is driven by a 210 H.P. engine, and the pumps by an 80 H.P. engine. The drums make 7½ revolutions per minute; the washed ore is discharged from the narrow ends direct into ore cars. The water from them runs into settling pits, the sludge from which is washed in two similar washing drums, but of half the size of the first ones. The overflow from the settling pits and the small washers is lifted by tailing wheels, 23 feet in diameter, into a couple of spitzkasten, about 10 feet square; the spigots of these discharge into two drums similar to the last, the overflow going to waste. The washed ore forms only 30 per cent. of the crude ore-clay.

Occasionally iron ores have been submitted to some form of wet crushing and dressing, usually by jigs, but at the present time iron ores are usually concentrated magnetically. Thus a good deal of the poorer magnetite, mined in the Lake Champlain district, used at one time to be crushed under tilt hammers, striking on cast iron gratings, down to about ¼ inch, the coarser portion jigged in Harz jigs, and the finer portion washed in a box buddle or a tye, the crude ore producing about 38 per cent. of concentrated ore.

At present this work is performed by magnetic separation. In one large works in the above district dry magnetic separation is employed. The ore comes from the mines in 8-ton hopper cars, which are hauled up an incline, and the ore is dumped into a main hopper which feeds two rock-breakers arranged in tandem, the first, a 30 inch by 18 inch Blake, breaking to about 3 inch, the second, a 36 inch by 6 inch double jaw Blake, to about 1 inch cube, the broken ore being conveyed from the first breaker to the second by a Robins belt. Another similar belt takes the broken ore from the smaller rock-breaker to a pair of 36 inch by 14 inch rolls, which crush it down to about No. 6 mesh. The crushed ore is elevated to the top of a drying tower, about 5 feet square in inside section, containing 48 rows of T-shaped bars, about 6 inches wide, 6 bars in a row, uniformly spaced; these bars are arranged in 8 sections of 6 rows each, all the bars in one section being at right angles to those in the sections above or below. The tower is fired by a lateral furnace. The dried ore runs out at the bottom of the tower and is elevated to the top of an inclined fixed screen, fitted with punched plates, making four sizes, namely 30, 16, 10 and 6 mesh. The oversize from the last or largest size goes to a pair of 36 inch by 14 inch rolls set over the conveyor belt that runs from the dryer to the boot of the elevator to the screens. Each of the four sizes goes to a Ball and Norton belt separator (see page 405) on to the belt of which it is fed by a roller feeder. The belts are 9 inches wide and can treat 20 tons per hour

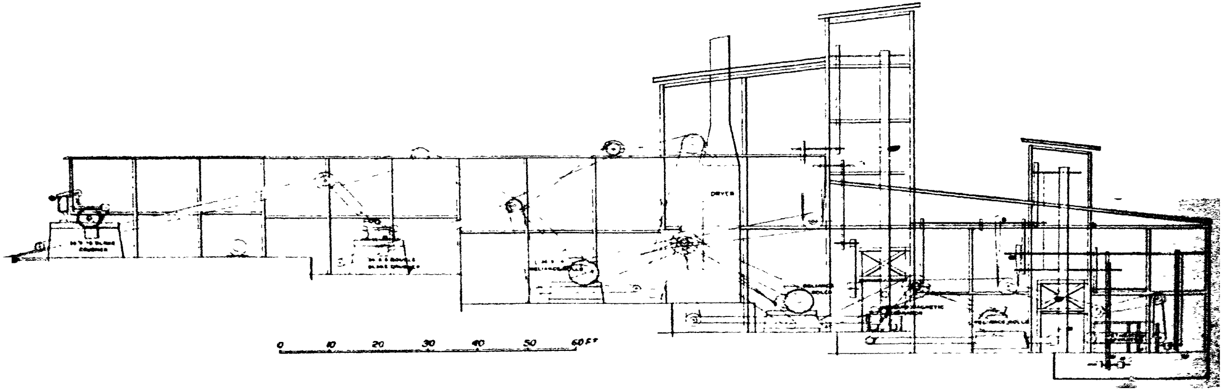

Fig. 399. Lake Champlain magnetic separator plant. Sectional elevation.

The tailings from the machines treating the finer sizes are crushed in another pair of 36 inch by 14 inch rolls, and the product goes to a 6-pole Wetherill machine (see page 430), the first pair of poles taking out magnetite, the second magnetite and hornblende middlings, the third hornblende, leaving marketable tailings of apatite, containing about 50 per cent. of phosphate of lime and 5 per cent. of metallic iron.

The final result is the separation of the crude ore into magnetite 83 per cent., apatite 7·5 per cent., hornblende, &c., 9·5 per cent.; the crude ore contains about 59·5 per cent. of iron and 1·75 per cent. of phosphorus, and the magnetite concentrate about 65 to 67 per cent. of iron and 0·5 per cent. of phosphorus. The mill treats about 750 tons of crude ore in 10 hours, and is worked by three 60 H.P. and one 10 H.P. induction motors, the average power consumption being 120 to 130 E.H.P. There are about 10 men engaged per shift.

Fig. 399, from a pamphlet issued by the owners, Messrs Witherbee, Sherman and Co., gives a general idea of the arrangement of the plant, but shews an older arrangement of the magnetic separators than the one described above.

A small plant has recently been erected by the Humboldt Engineering Works Co. at Eiserfeld, Sieg, to treat calcined spathic iron ore, the separator used being of the type described on page 396. A section of the installation is shewn in Fig. 400. The ore is hoisted by an ordinary hoist and tipped into a bin *A*, whence a simple reciprocating feeder *B* feeds it uniformly into the trommel *C*, having screens of 1·25 inch, 0·6 inch and 0·24 inch mesh respectively.

The oversize from the largest screen drops down a shoot on to a rotating picking table *D*; on this table some 8 or 9 tons are picked over per 10 hour shift, 4 or 5 lads being ample for this work. They pick out waste and clean lump ore; the remainder goes to the rock-breaker *E*, and after breaking is returned to the hoist. All the stuff that is between 1·25 and 0·6 inch in size goes through a pair of crushing rolls, and after being crushed is also sent up again by the hoist. The undersizes from the two finer screens go each to a magnetic separator F_1 and F_2; the middlings from these go to a pair of crushing rolls, and after crushing pass through a third separator F_3, the middlings from which are returned to the last-named crushing rolls. The entire plant is driven by a 45 H.P. motor, and can treat in 10 hours 60 tons, containing 30 per cent. of iron; the concentrates amount to about 29 tons and contain 56 per cent. and the tailings 16 per cent. of iron. The cost amounts to about 4s. per ton of product.

As an example of a wet magnetic separating plant, a small Scandi-

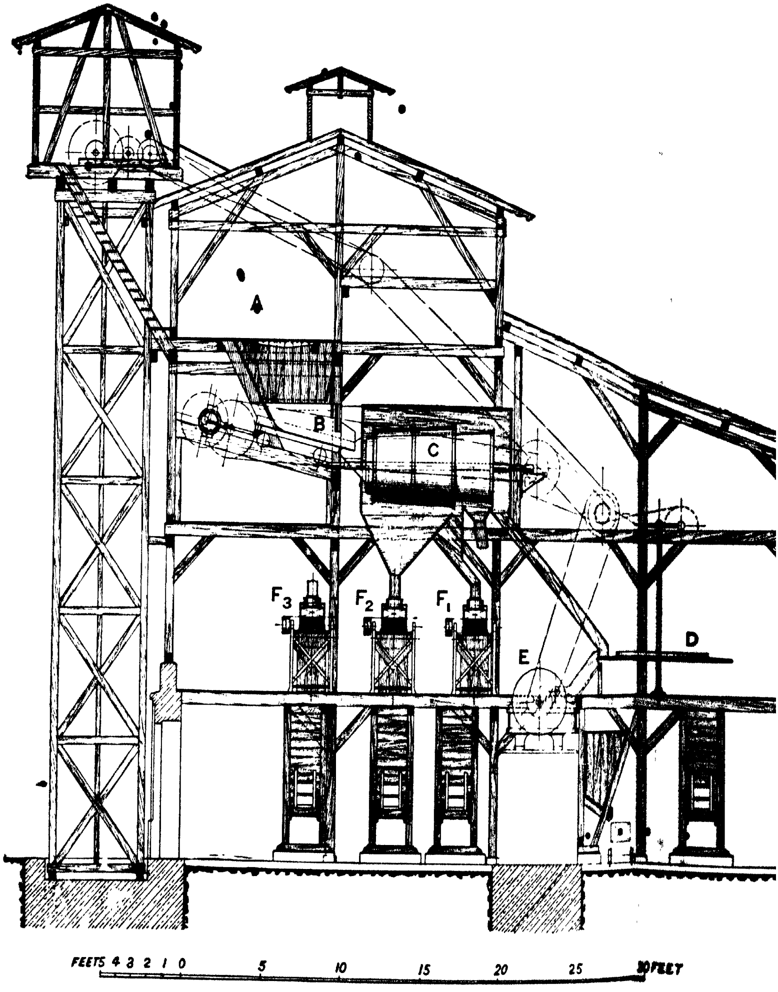

Fig. 400. Sieg magnetic separator plant. Sectional elevation.

navian plant[1] may be quoted, which is shewn in Fig. 401. The ore comes in direct from the mine in side-tipping cars, and is thrown into a Gates rock-breaker, from which it drops into a shoot feeding into two Gröndal Ball mills, making 28 revolutions per minute; these mills were originally charged with 50 balls weighing 26 lbs. and 50 balls weighing 13 lbs. each; on an average each mill requires 6 new 26 lb. balls per week. The outflow ends of these mills are fitted with conical screens of 0·08 inch mesh, the water consumption being altogether about 110 gallons per minute. The pulp flows to four Gröndal separators of No. 2 type (see page 423), making 40 revolu-

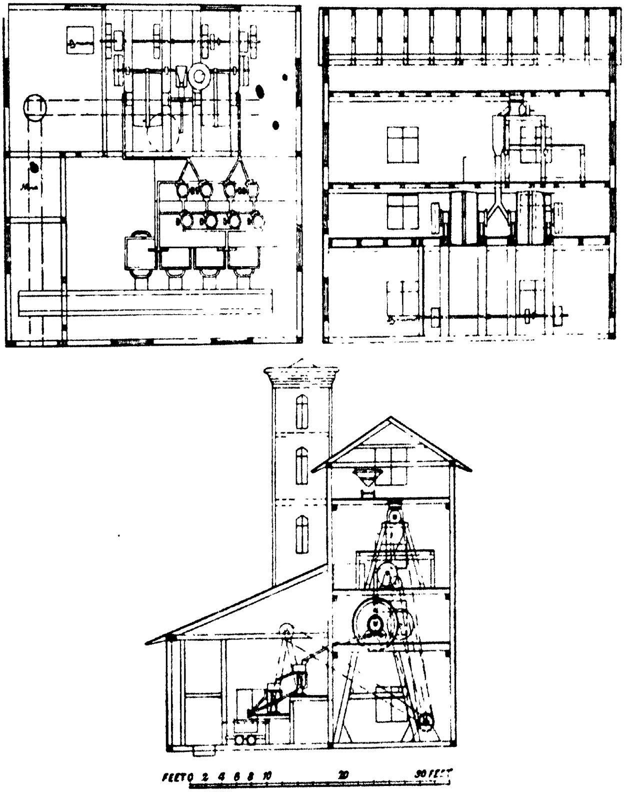

Fig. 401. Scandinavian wet magnetic separator plant. Sectional elevation. Plan of lower (concentration) floor. Plan of upper (Ball mill) floor.

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[1] *Jernkontorets Annaler*, 1903, p. 312.

tions per minute. These produce tailings that run direct to waste. The magnetic portion goes to four similar, but less powerful, concentrators (the magnets of the former are wound with 4800 turns, those of the latter with 1820 turns), which produce middlings, which are run into cars; the latter are elevated by a hoist, and the middlings thus returned to the ball mills for further crushing. The concentrates fall on to an endless revolving screen with 0·02 inch mesh; the oversize from this drops into a car and is also hoisted up and returned to the ball-mills. The undersize forms the finished product. The crude ore contains about 47·4 per cent. of iron and 0·72 per cent. of phosphorus; the concentrate about 68 per cent. of iron and 0·119 per cent. of phosphorus; the tailings contain 7·1 to 8·75 per cent. of iron, of which 3 per cent. is in the form of magnetite. The works treat 90 tons of crude ore per 24 hours and produce about 40 tons of concentrates; there are two 12 hour shifts with 8 men per shift; the power consumption is 55 H.P.

In the newer plants a rather more elaborate arrangement is preferred. The ore is broken, usually in tandem rock-breakers, the first being usually a jaw breaker and the second of the gyratory type. After each breaking, or after only one of them, according to the character of the ore, barren rock or mineral too poor to treat may be sorted out, usually on picking belts; at times this sorting is performed magnetically by a magnetic separator of the Wenström type. The ore to be treated is then fed into wet crushers, usually ball mills, where it is crushed to $\frac{1}{10}$ or $\frac{1}{4}$ inch. The pulp then passes to magnetic separators of such strength as to produce practically clean tailings; the magnetic portion is fed into a tube mill where it is further crushed to the requisite degree of fineness, sometimes to $\frac{1}{100}$ inch, and this fine pulp goes to another set of magnetic separators, best so arranged as to give rich magnetic concentrates, clean tailings and middlings that are returned to the tube mill for re-crushing.

A fine plant on similar principles has been erected at Stråssa, in Central Sweden, by the Metallurgiska Aktiebolag of Stockholm, and is shewn in Fig. 402. The ore to be treated consists of a mixture of magnetite and specular haematite in about equal proportions in a gneissose gangue, the ore averaging some 35 per cent. of metallic iron. The ore is delivered to the storage hopper **A**, whence it drops into a 25 inch by 16 inch Blake rock-breaker **B**. The broken ore is lifted by the Robins belt conveyor **D** into a pair of Gates rock-breakers **C**, 36 inches in diameter. Thence another Robins belt conveyor takes the ore, now broken to about $\frac{3}{4}$ inch, to hoppers from which four revolving ore-feeders **F** feed it into four Ball mills **E**. The pulp runs into Gröndal tandem separators **G** of Type 3 (see p. 423). The magnetite thus separated

is re-ground in the 4 tube-mills **J**, and after grinding is finally concentrated on similar Gröndal separators. All the tailings from the magnetic separators goes to a pair of four-compartment spitzkasten **K**, the product from each spigot going to one of 8 Ferraris tables **L**, which give a clean haematite concentrate. The haematite and magnetite concentrates are mixed and go to the draining appliances **M**. These consist of boxes, triangular in vertical section, hung on a hinge at one end and shaken rapidly by a simple cam acting against a tappet at the other end. The contents of the box are thus kept in rapid vibration, under the effect of which they settle very closely, the water running off over the edge of the box. On inverting the box, the contents, now sufficiently freed from water, drop into small cars, which convey the concentrates to storage bins **O**. The concentrates are then stamped in briquetting presses of the drop-press type, **N**, and are burnt in Gröndal tunnel furnaces **P**, fired by gas producers **R**. A full description of this method of briquetting will be found in the *Journ. Iron and Steel Inst.*, Vol. LXV., 1904, p. 40. The briquettes are stored in bins **T**, whence they are transported to the railway by the aerial cable-way **U**. The plant will treat about 300 tons of crude ore, producing about 150 tons of briquettes in 24 hours. It is worth noting that it has been found in treating many ores of this type, that the specular haematite, when associated with magnetite, is sufficiently magnetic to be capable of being concentrated by wet Gröndal magnetic separators, using a more powerful field than suffices for magnetite.

A small plant in Central Sweden consists of a Blake rock-breaker 24 inches by 16 inches, which breaks the ore down to about 2 inches. The broken ore drops into a bin holding about 150 tons, from the bottom of which a roller feeder delivers it to a Robins carrying belt that takes it to the feeders of a couple of Ball mills, 6 feet 6 inches long by 6 feet 6 inches diameter, making 24 revolutions per minute. Each ball mill holds about 1 ton of balls, weighing 28 lbs. each; the wear amounts to about 165 lbs. of metal per 24 hours. The two mills crush together about 140 tons in the same time, and require about 100 H.P. to drive them. The water supply is about 300 gallons per minute. The pulp from the ball mills flows to a Gröndal magnetic spitzkasten, followed by a pair of Gröndal separators of the No. 3 type (see p. 423) working in tandem. The tailings flow to waste. The concentrates, which form about ⅓ of the weight of the ore, go to a tube mill 13 feet long by 4 feet in diameter, making 26 revolutions per minute, and using a charge of 3 tons of Danish quartz pebbles; apparently one such charge will grind about 5000 tons of concentrates. The finely ground pulp is lifted by a bucket elevator and concentrated in a pair of Gröndal separators with magnetic spitzkasten as before. The concentrates are unwatered in a shaking appliance and are then briquetted. The concentrates contain about 67 per cent. of metallic iron. The magnetic separators all together take a current of 80 volts at 80 amperes.

Manganese ores are not often the subject of dressing operations; sometimes they occur with much adherent clayey matter, and in that case have to be washed like brown haematite. Other ores usually require only hand-picking. In a few cases the ores are broken down by rolls and dressed in jigs, the finer material being buddled or treated on Wilfley tables. In small mines the ores are often washed by hand in tyes or strakes.

Copper ores. Oxidised ores of copper are rarely dressed, as they are usually treated by metallurgical or chemical means; it would be quite practicable to dress an ore containing say cuprite or malachite, but owing to the brittleness of these minerals they would be in great part reduced to fine slimes, and the losses in dressing would be heavy.

Native copper is dressed on a large[1] scale in the important mining district of Lake Superior. The typical method of treatment there is as follows:

The ore goes first to spalling floors, where it is broken up and the large lumps of native copper picked out; this breaking is often supplemented by jaw rock-breakers, and steam hammers for freeing the copper as far as possible from adherent rock. The rock to be treated then passes to the steam stamps; usually each steam stamp, and the dressing plant combined with it, are treated as an individual unit, which may be stopped or started independently, and a mill consists of the requisite number of such units. The steam stamps have screens with holes about $\frac{3}{16}$ inch in diameter. The coarse copper remaining in the mortar is picked out at intervals; the pulp runs to a set of 4 hydraulic trough classifiers, each with 4 spigots. The product of each spigot goes to a jig, generally a Collom jig (see p. 286). The tailings from these run to waste; the concentrates are practically ready to send to the smelter. The middlings from the intermediate hutch go to other jigs or else to Wilfley tables. The overflow from the classifier and at times some of the finer products from some of the other dressing machines go to 4 or 5 ordinary convex slime tables. There is rarely any re-crushing of middlings, as the smelters can treat comparatively low grade stuff, and the low percentage of copper in the crude ore renders rapid and cheap working essential. The crude ore may contain about 2 per cent. of copper, and the dressed material will average about 80 per cent. Each unit will treat about 350 tons per 24 hours; the steam stamp absorbs about 100 H.P. and the washing plant, etc., about 13 H.P. or say 113 H.P. altogether, exclusive of pumping; the water consumption is about 100,000 gallons per hour.

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[1] *Ore Dressing*, R. A. Richards, Vol. II. p. 990.

This description may be illustrated by the single-unit plant for treating 125 tons per day of 24 hours, built by Messrs Fraser and Chalmers, Ltd., and shewn in Figs. 403 and 404. Here *A* is a grizzley and *B* a 15 inch by 9 inch Blake rock-breaker; the ore from these goes to the steam stamp *C* with 26 inch by 11 inch cylinder, stamping through $\frac{3}{16}$ inch screens. The crushed pulp goes to hydraulic separators *D*, making three products that go to three pairs of Collom jigs *E*, *E'*; the middlings from these jigs are re-crushed in Heberle or in Huntingdon mills *F*; the products go to a second series of hydraulic separators *G*, the first two spigots giving products dressed on the Collom jigs *H*. The slimes from all the plant run into the large spitzkasten *J*, the 6 spigots of which supply 6 Frue vanners *K*. In case of need a second row of Frue vanners *K* can be put in to complete the treatment of the slimes. In this case it would probably be preferable to replace the first row of Frue vanners by Wilfley tables.

Ores containing copper pyrites or cupriferous iron pyrites frequently require dressing, and the treatment of such an ore may be taken as typical of the majority of dressing operations. Usually these ores occur in a quartzose gangue and the copper and sulphur contents are very irregularly distributed. A certain proportion of lump ore can usually be picked out that is saleable as such or can be smelted direct; another portion will generally consist of waste, or of vein stuff containing too little copper to be worth treating; the remainder will have to be dressed, and in the best modern practice will be treated by the system of gradual reduction. The following is a description of a typical works for the treatment of such ores, which is illustrated by Fig. 405.

The ore comes in from the mines by means of an aerial ropeway; the buckets pass over an automatic registering weighing machine, and are tipped by hand on to one of four grizzlies with bars 3 inches apart; the coarse ore is trammed to a rock-breaker, with jaws about 24 inches by 16 inches, any lumps of shipping ore being put aside; the broken ore drops into a bin, whence a shoot furnished with a sliding gate delivers it on to a pair of belts 2 feet wide, 50 feet long, travelling at the rate of 50 feet per minute, there being 4 to 6 pickers to each belt. Lump shipping ore and waste are picked off here, whilst the dressing ore

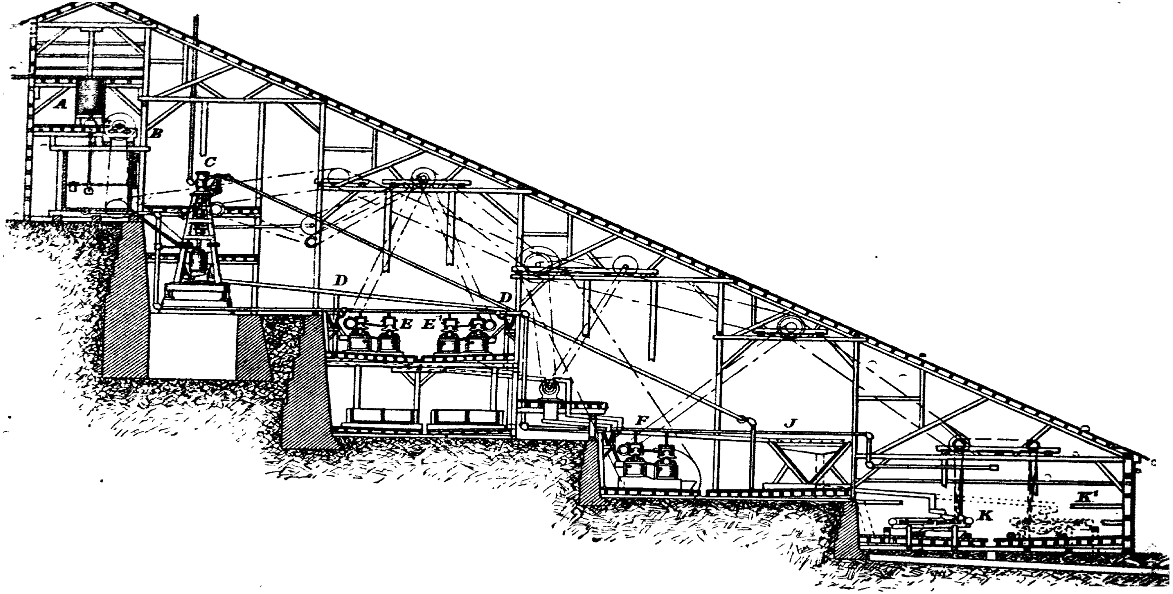

Fig. 403. Steam stamp plant for native copper. Sectional elevation.

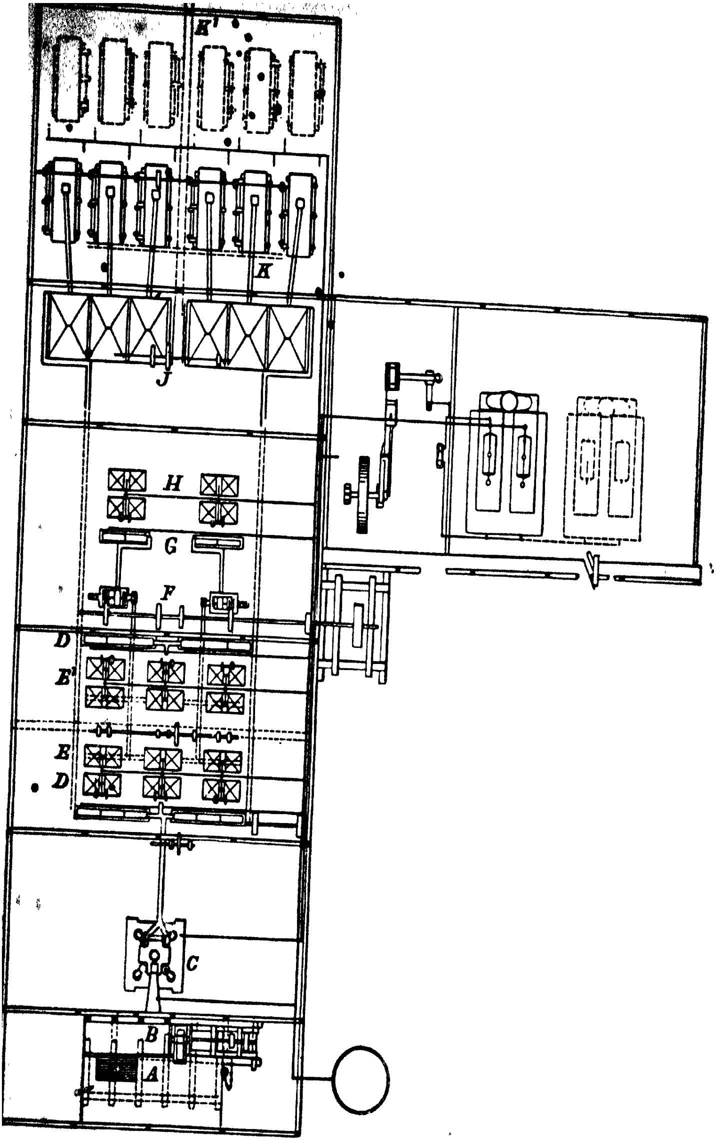

Fig. 404. Steam stamp plant for native copper. Plan.

mixing with the undersize from the grizzlies is carried by a pair of Robins belt conveyors into the dressing works proper, the operations previously described taking place in the sorting house, which here forms a separate building. It is the dressing plant alone that is shewn in Fig. 405. Each belt feeds into a pair of trommels of punched sheet iron with 1-inch holes; the oversize goes to a pair of picking belts, with 8 pickers to each belt, who pick out waste rock and lump ore, leaving the dressing ore on the belt. These belts deliver into a bin from which a rock-breaker with jaws about 18 inches by 6 inches is fed by an automatic feeder. From this point forward the mill is built in duplicate, each half being exactly the same. The undersize from the above trommels mixes with the ore broken in the rock-breaker, and passes to four trommels with 0·4 inch holes, the oversize from each of which goes to a pair of rough crushing rolls. The undersize from each trommel goes to a set of six sizing trommels with apertures respectively of 0·28 inch, 0·20 inch, 0·16 inch, 0·12 inch, 0·08 inch and 0·04 inch. Each trommel delivers its product to a three-compartment Harz jig, the bed consisting of clean pyrites somewhat larger than the mesh of the jig screen.

The following table shews the details of the action of these jigs:

For stuff passing trommel of mesh	Number of strokes per minute	Length of piston stroke for		
		1st Compartment	2nd Compartment	3rd Compartment
inch		inch	inch	inch
0·40	130	1·60	1·50	1·42
0·28	140	1·42	1·34	1·18
0·20	150	1·10	1·02	0·90
0·16	160	0·87	0·79	0·71
0·12	170	0·79	0·71	0·59
0·08	180	0·63	0·55	0·47
0·04	200	0·55	0·47	0·35

The spigot of the first compartment of each jig gives clean coarse pyrites, the second also clean pyrites of rather smaller grain, the third gives mostly middlings, whilst the overflow is collected and delivered to the lower works, or slime dressing plant. The tailings from the jigs treating the four finer sizes go direct to the lower works, the tailings and middlings from the three jigs treating the coarser stuff go to a pair of re-crushing rolls; the re-crushed material is sized by sets (two in each half of the mill) of three trommels with 0·12 inch, 0·08 inch and 0·04 inch mesh, with a small square spitzkasten set below the lowest one; the products from

these go each to one of four three-compartment jigs, which make clean pyrites; the tailings from the last three jigs go to the slime plant, those from the coarsest one to the last set of rolls. Thus all tailings or middlings finer than 0·08 inch go direct to the slime plant, and all coarser are crushed down to this size.

Each half of the upper works delivers its products to its own classifying trough (see p. 236), this being 34 feet 6 inches long, widening from 3 feet 9 inches to 7 feet 4 inches, and from 3 feet 3 inches to 6 feet 6 inches deep, divided into 5 compartments. The first spigot supplies two sets of jigs, the second one set, and the three last a pair of Lührig vanners each. The tailings from these last go to waste, and the middlings together with the tailings from the jigs go to a classifying trough 62 feet long, increasing from 3 feet 3 inches to 9 feet 10 inches in width, and from 2 feet 2 inches to 6 feet 6 inches in depth, divided into 7 compartments. The first spigot delivers to a 4-compartment jig, the next to a 3-compartment jig, and the other 5 spigots to Lührig vanners, there being altogether 14 Lührig vanners in each half of the slime dressing plant. The overflow from the head boxes of these 28 Lührig vanners goes to a labyrinth in which a good deal of rich floating cupriferous pyrites is collected. The entire works are driven by two 100 H.P. turbines, and their capacity is 1000 tons of crude ore per 24 hours; of this quantity 200 tons is picked out as lump ore and waste, and the remaining 800 tons are dressed. The water supply to the works is about 1500 gallons per minute; there are about 75 pickers employed in the sorting house and 30 men in the dressing works per shift.

This plant furnishes an excellent example of gradual reduction, and may be taken as typical of wet dressing plants in general, although of course the individual appliances employed are subject to a very wide range of variation. The tailings from such a works as that here described will usually carry at least 0·3 per cent. of copper, when the crude ore carries about 3 per cent. In modern practice tables of the Wilfley type are employed for the coarser grades in slime plants, and vanners or revolving tables of the Harz type or some modification thereof only for the finest portions. The Elmore vacuum process promises to be very successful in saving copper from the slimes of such ores, and if the favourable results hitherto obtained are confirmed by more extended experience, such plants will in the future be built like the above as far as the jig mill is concerned, to be followed by tables of the Wilfley type and these by Elmore vacuum machines.

Lead ores. Practically the only lead ore that the dresser is called upon to treat is galena, which occurs for the most part in veins, associated with quartz, calcspar, fluorspar, spathic iron ore, etc., and in some cases also with zinc blende; this latter mineral will be referred to subsequently.

As an example of the ordinary method of treating such lead ores, the dressing mill of a North Country lead mine may be quoted.

The crude ore or "work," containing about 5 per cent. of galena[1], is trammed out of the mine in waggons or hutches, and tipped into a hopper, from which it is raked over a grizzley with bars about 2½ inches apart, a stream of water playing upon it; any waste not worth dressing is here picked out and thrown aside, and also any lumps of pure galena, known as "potter ore"; the remainder or "bouse" is carried by a short belt to a rock-breaker with jaws 16 inches by 5 inches, which breaks it down to 2 inches. The broken stuff, together with the undersize from the grizzley, drops direct on to two pairs of rolls, set one immediately above the other, intended to break it all to ⅜ inch. The broken stuff is elevated to a set of 5 conical trommels having meshes respectively ⅜ inch, $\frac{5}{16}$ inch, ¼ inch, $\frac{3}{16}$ inch and ⅛ inch wide. The oversize from the first goes back to the crushing rolls, and the undersize from the last to the slime plant. The sized stuff goes to four 3-compartment Harz jigs, with beds of galena, which give clean galena in the first compartment, middlings or "chats" in the second, and waste tailings or "cuttings" at the overflow; the third compartment gives material that is often clean enough to go to waste, but at other times is treated as middlings.

The middlings are re-crushed in a special pair of rolls, known as "chat rolls" and the crushed stuff is elevated to the main trommels. The degree of crushing and the quantity of material to be re-treated are left to the judgment of the foreman of the washing mill. The slime plant consists of 3 ordinary round buddles for the coarser stuff and 2 Brunton tables for the finer; the tailings are settled in 3 strips and dug out from these from time to time, and washed on 4 more Brunton tables, this latter part of the work being done by a contractor. The heads from the Brunton tables are dollied in hand dollies, those from the round buddles are finished on a knife buddle, whilst the clean ore from the jigs is finally dressed upon a small flat table worked by hand. The product is galena with 77 to 80 per cent. of lead, whilst the tailings appear to contain about ½ per cent. of lead, though systematic tailing assays are not often made. This mill produces about 12 to 15 tons of dressed ore per day, or treats about 250 tons in a 10 hour shift, about 8 men being employed.

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[1] In the lead mines of the North of England "work" is usually spoken of as giving so many "bings" of dressed ore to the "shift" of "work," a bing being 8 cwt., and a shift 120 cwt.

A more elaborate plant[1], shewn diagrammatically in Fig. 406, is arranged as follows: The ore as it comes from the mine is tipped into a hopper, whence it passes over 2 grizzlies set one above the other, the upper with bars about 1½ inches apart, the lower with bars about ½ inch apart. The oversize from the top grizzley is hand picked, any waste being removed, and the remainder goes to the rock-breaker. The oversize from the second grizzley goes to the crushing rolls, the undersize to the trommels, whilst the escaping water deposits any fines it may carry in settling tanks. A stone-breaker of the Blake type, driven by a 15 H.P. turbine, breaks the coarser stuff to 1½ inches, and this together with oversize of the finer grizzley passes to the crushing rolls, consisting of one pair of grooved rolls 16 inches in diameter by 17 inches long, from which it goes to two pairs of finishing rolls, of the same size but smooth, the former making 8 and the latter 16 revolutions per minute; these rolls crush about 4 or 5 tons per hour. The crushed stuff passes through the trommel *A* with 2½ square meshes to the inch, the oversize being taken to a pair of rolls for finer crushing. The undersize passes to a set of 3 trommels *B*, *C* and *D*, having respectively 3, 4 and 6 holes to the inch, the undersize from the last going to a set of 4 spitzkasten *E*. The oversize from each trommel and the products from each spitzkasten go to a separate Harz jig (Nos. 1 to 7), the details of which are given in the following table:

Jig Number	Number of compartments	Strokes		Nature of grid
		Number per minute	Length	
			inch	
1	4	140	½	Steel sheets with ⅜ inch round holes
2	4	140	½	" " " ¼ " " "
3	8	180	⅜	Copper sheets " No. 7 " "
4	8	200	5/16	" " " " 14 " "
5	8	220	¼	" " " " 18 " "
6	8	230	⅛	" " " " 28 " "
7	8	250	⅛	" " " " 29 " "

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[1] *Trans. Inst. Min. Eng.*, "Description of the Lead-Ore Washing Plant at the Greenside Mines, Patterdale," Vol. xxv. 1902, p. 331.

All the jigs produce good galena in the first compartment, that from No. 1 jig assaying about 75 per cent. of lead, and that from all the others about 80 per cent. The second compartment produces galena with barytes, pyrites, and some blende; the third (and fourth) yield poor middlings consisting of vein stuff with specks of galena; whilst the overflow is considered to be waste tailings.

The overflow from the spitzkasten *E* passes down to the second floor of the works, where it enters a large spitzkasten, *G*, the spigot from which feeds two large round buddles *H* 1 and *H* 2, 24 feet in diameter with headboards 8 feet in diameter; the overflow from the spitzkasten goes to another spitzkasten *I*, supplying two similar buddles *H* 3 and *H* 4. The contents of the buddle are divided into 4 cuts; the first is dressed up to standard quality on a small buddle, the second and third are buddled again in another small buddle, and the tailings are run to another classifier and buddle. The oversize from the trommel *A* is trammed to rolls on the second floor and the crushed stuff is sized in two trommels followed by two spitzkastens, the various sizes being jigged in four jigs, 1^A, 2^A, 3^A and 4^A. The ore from the first compartments of all the jigs is cleaned in a special cleaning jig or on a flat table or "trunking-box," producing clean galena with 82 to 84 per cent. of lead.

The rich middlings from the second compartments of jigs 1, 2 and 3 is trammed to the chat-rolls or No. 3 crusher, classified and jigged in two jigs *S* and *T*, the former running at 200 one-half-inch strokes per minute, the latter at 280 quarter-inch strokes. The first compartment yields practically clean galena, and the middlings are treated with the poor middlings from the other jigs. These are in part stamped in a small stamp mill, in part crushed in the No. 4 crusher rolls, and the product is classified, the rougher portion going to a jig making 280 strokes of $\frac{1}{16}$ inch per minute, the finer being treated on Lührig tables. Below these come settling pits, the slimes from which are further treated on round buddles. The crude ore from the mine carries about 7 per cent. of galena, which is brought up by hand-sorting to about 15 per cent. The produce of the entire mill is about 6 tons of lead ore per 10 hours' day and the cost of dressing about 10*s.* 6*d.* per ton of dressed ore.

A modern plant for treating an ore of argentiferous galena, which is very easily dressed, has been described by Mr E. R. Woakes[1]. The ore consists of galena in a gangue of quartz with smaller proportions of pyrites, magnetic pyrites, zinc blende, etc., which are too low in silver

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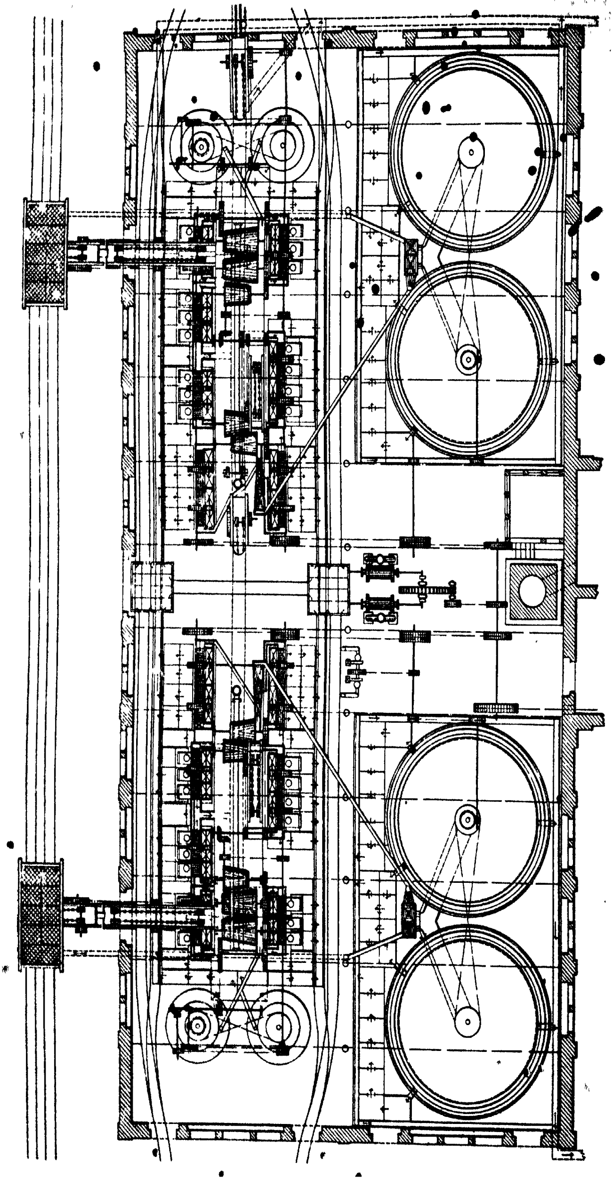
[1] *Trans. Inst. Min. Met.* Vol. XII. 1902-3, p. 140.

to be worth saving. The scheme of operations is shewn diagrammatically in Fig. 407, such a diagram being generally known as a "Flow sheet," whilst the plant itself is shewn in Fig. 408. The ore is delivered by an aerial tramway to bins, from which it passes over a grizzley to a Gates rock-breaker, and the broken ore is carried by an 18 inch belt conveyor to a storage bin. From this it is fed by an automatic feeder to the coarse crushing rolls, 26 inches diameter by 15 inches face, running at 85 revolutions per minute. The broken ore drops into a trommel 36 inches in diameter and 40 inches long with $\frac{7}{8}$ inch round holes. The oversize goes to the intermediate rolls, the undersize to an elevator consisting of steel buckets 12 in. by 6 in. attached to a 14 inch belt; the latter runs at about 350 feet per minute, the distance between head and tail pulleys being 48 ft. 6 in. The crushed ore goes to a set of 4 trommels all 36 in. in diameter; the first has $\frac{1}{2}$ in. and $\frac{11}{16}$ in. holes, the second has $\frac{1}{4}$ in. round holes, the third 0·12 in. round holes and the last 16 mesh slot holes. The oversize of each mesh goes to one of 7 jigs, there being 2 two-compartment, 3 three-compartment and 2 four-compartment jigs, the screens in each compartment being 3 ft. by 2 ft. The four-compartment jigs are fed from classifiers. The undersize from the last trommel goes to a set of 4 hydraulic classifiers and thence to the V-shaped settling boxes, the first having three divisions. There are thus 8 spigots, the first two of which feed to the two jigs, and the remainder each to one of 6 Wilfley tables.

The first compartment of the jigs and the tables produce ore clean enough for shipment; the jig tailings are mostly clean enough to go to waste, those from the first at times, however, carrying enough ore to make them worth re-crushing. The middlings from the coarser jigs go to the intermediate rolls, those from the finer jigs to the fine crushing rolls, of the same size as the coarse rolls, but running at 95 and 105 revolutions per minute respectively. The crushed material goes back to the boot of the elevator.

The mill is driven by a 4-foot Pelton wheel under a head of 450 feet, and its capacity is about 200 tons per 24 hours. Its total cost complete was just about £7200, and the cost of treating the ore just about 15*d.* per ton. The saving of lead amounted to 81·5 per cent.

Lead-zinc ores. As galena and zinc blende occur together in a large number of deposits, and as the presence of either mineral in notable quantity interferes with the metallurgical treatment of the



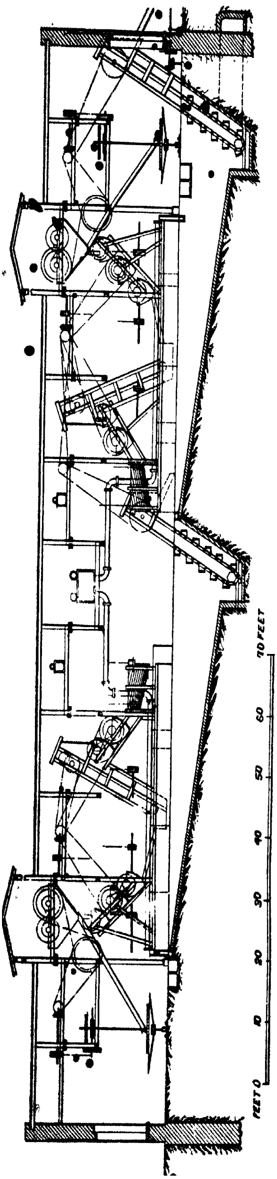


Fig. 409. Lead-zinc dressing plant. Plan and longitudinal section.

other, the problem of dressing such mixed ores is one of the most commonly occurring in ore-dressing practice. When the mixture is very intimate and the minerals excessively fine-grained, the problem presents extreme difficulties and can hardly be said even now to have been entirely solved; in such cases (e.g. at Broken Hills, New South Wales) magnetic methods, chemical methods and more recently flotation methods have been employed. These are examples of the use of individual appliances or processes for a special operation, and have been sufficiently treated under the heads of the respective appliances. It is only necessary here to consider complete plants using the ordinary wet methods for the separation of these minerals when they occur in particles large enough to admit of wet separation.

Such a plant, erected in Germany, for treating an ore consisting of blende and some galena in a gangue of quartz, slate, spathic iron ore, etc., is shewn in Fig. 409[1]. It will be seen that the right-hand and left-hand halves of the works are quite identical in design and the following description refers to one half only. The ore comes from the breaking house, where it has been broken small, by the main tracks, in mine waggons, which are

[1] Linkenbach, *Aufbereitung der Erze*, Pl. XXII.

tipped on to the gratings: a system of bucket elevators on chains, lifts the ores to a couple of washing trommels, with holes 1·2 inches in diameter, the oversize of which goes to a pair of picking tables; the dressing ore is sent back to be re-crushed. The undersize goes to a set of five conical sizing trommels, arranged in two groups to save head room with an elevator between them. The apertures are of 0·7 inch, 0·4 inch, 0·2 inch, 0·12 inch and 0·06 inch widths respectively; the undersize goes to a small two-compartment spitzkasten, the overflow from which goes to the slimes washery. Harz jigs treat separately the product from each trommel, the three coarser sizes going to 3-compartment, the two finer to 4-compartment jigs. The sands from the spitzkasten are treated on two 4-compartment jigs. The overflow from this spitzkasten passes to another spitzkasten, which supplies two Linkenbach tables; the overflow from these runs directly to waste through a launder. The middlings from all the jigs runs into settling pits, whence they are loaded into waggons running on the tracks, and taken to be re-crushed. The waste runs down inclines and is lifted by elevators till it reaches the endless chain haulage, which takes it to the waste tip. There are two pumps, pumping the requisite water into high level tanks, whence it is distributed by cast iron pipes.

The entire works can treat 15 tons of broken ore per hour; it is driven by a 36 H.P. steam engine, requires 660 gallons of water per minute, and employs 80 hands, of whom 40 are engaged in picking.

This is a fair example of a somewhat old-fashioned plant, in which all the crushing machinery is kept distinct from the dressing plant proper, this being somewhat contrary to the usual practice.

A modern German mill for treating similar ores is shewn in Fig. 410[1], which consists of three identical divisions, the whole works being capable of treating 30 tons per hour. The ore from the mines is brought at the level *a* in mine waggons into the building *C* where it is tipped on to the grizzley *b*; the oversize drops into the bin *c* whence it is taken to the sorting house. The undersize passes into the bin *d*, whence it is fed into the washing trommel *e*; the undersize is carried by the shoot *k* to the first sizing trommel *m*. The oversize is sorted on the picking belts *f*, on which waste and lump ore are picked off and thrown into shoots *n*, from which they are readily loaded into waggons. The dressing ore drops into jaw rock-breakers *g*, and thence into trommels *h*; the undersize from this joins that from the first trommel. The oversize is again picked on round picking tables *i*; lump ore and

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[1] C. Blömeke, *Zeitsch. f. B. H. u. Sal.-Wesen.* 1904, B. p. 17.

waste is again thrown into shoots, and the dressing ore goes to the coarse-crushing rolls l, the crushed ore going to the same trommel m, which receives the undersize from the upper trommels. The sizing plant consists of the four trommels m, m^1, m^2, m^3, and a small spitzkasten o. The material from the first trommel is jigged in three 4-compartment jigs p, whilst that from the other three trommels and the spitzkasten is distributed to ten 5-compartment jigs q and q'. The clean products from these jigs is dropped into shoots r whence it is taken away in cars, whilst the middlings go to two pairs of re-crushing rolls s; the tailings from all the jigs run down to the boot of an elevator t, which lifts them into the waste hopper u. The crushed middlings are lifted by the elevators v to a pair of trommels w followed by spitzkasten x; the products of these are jigged in the five sets of 5-compartment jigs y, the clean ores from which are dropped into the shoots z and thence into cars.

The overflow from all the three divisions now passes into the slimes washery B and runs into a series of large spitzkasten a_1, in which the slimes are deposited and further thickened in the spitzkasten b_1, the various spigots of which supply the four Bartsch round shaking tables (see page 371) and the three Lührig belt vanners d_1. The middlings from these 7 machines run to the spitzkasten e' and are dressed on the three Bartsch tables f_1 and the three Lührig vanners g_1, the middlings from which collected in No. 1 are finally dressed again on the Bartsch table i_1.

This may be looked upon as a typical modern plant in which the principle of gradual reduction is well carried out, but in which the ore is so granular as to form but little slimes; as a rule the slimes washery is larger in proportion to the rest of the dressing works than is the case in this instance.

Zinc ores. The treatment of blende, in its most general mode of occurrence, has just been considered. When an ore contains little or no lead ore with the blende, it is dressed just as a lead ore would be. Difficulties arise at times owing to the comparatively low specific gravity of zinc blende, on account of which it is difficult to separate from barytes, or from spathic iron ore, or from iron pyrites. Most of the first-named can usually be got rid of by careful picking. The separation from spathic iron ore is usually performed magnetically, either by lightly roasting the ore and removing the magnetic oxide of iron thus formed by some form of magnetic concentrator for minerals of high magnetic susceptibility, or by treating it dry but unroasted by the Wetherill process or some analogous machine, so as to remove the spathic iron ore, which though of low magnetic susceptibility, is yet more magnetic than most zinc blende. A very ferruginous blende is however often found to be so magnetic that this latter method cannot be employed, and then the former one must be used.

Again zinc blende has often to be separated from iron pyrites; this is also done magnetically, a very dark ferruginous blende being often more magnetic than the iron pyrites, when the Wetherill machine can be applied with advantage. When the blende is not sufficiently magnetic, the mixture may be very lightly roasted, so as to convert the iron pyrites into magnetic pyrites, or else it may be more strongly roasted so as to convert the iron pyrites into magnetic oxide and the bulk of the zinc sulphide into oxide; in either case the magnetised iron mineral can be separated readily by magnetic means from the ore of zinc. It is proper to observe that both spathic iron ore and iron pyrites are injurious in the metallurgical treatment of zinc ores.

Calamine often occurs in a very tough clay and then needs washing, exactly as in the corresponding case of brown haematite.

The problem, already referred to, of separating very finely disseminated and intimately intermixed blende and galena, such as occurs in the Broken Hill district of New South Wales is a highly difficult one, and in this case it is further complicated by the presence of rhodonite, garnet, etc. Wet concentration and magnetic concentration in various forms have been tried without success; at present better results seem to be attained by the use of flotation processes, but it cannot yet be said that a definite solution has been reached.

Tin ores. Alluvial tin ores usually need merely washing in an ordinary trough or tye; owing to the great weight of the tinstone and its well marked granular form it is very easily concentrated practically without loss.

Tin ore occurring in veinstuff presents on the contrary one of the most difficult problems with which the ore-dresser is called upon to deal. The gangue is usually quartz, felspar, fluorspar, and other similar minerals together with iron, copper and arsenical pyrites, and very often wolfram, whilst all these minerals occur minutely disseminated, the tinstone forming perhaps 1 per cent. of the whole mass.

The old Cornish method was somewhat as follows: the ore from the mine is tipped from a high trestle on to floors or "slides," where it was submitted first to "ragging" or breaking with sledge-hammers, and then to "spalling" or breaking down to about 3 inch cubes; any pieces of copper ore, or of clean wolfram, or of barren waste, are next picked out, and the tin ore is wheeled to the stamps; it is tipped into the "pass" behind the stamps and by the aid of a stream of water is carried down into the cofers of the stamps and is there stamped fine, until it passes through the grates, which are generally punched with 144 holes to the square inch. The pulp flowing from the stamps is usually settled in "strips" 20 to 30 feet long, 18 inches wide and 18 inches deep, set at a slope of about 1 in 30, there being 3 strips to each battery box. In each strip, the "heads" settle first, then the "crazes" or middlings, and last the "tails," some slimes etc. escaping. The material thus obtained is buddled, each class by itself, on ordinary convex round buddles; the tails are shovelled out and carried to waste, the middlings are buddled over again, and the heads are further cleaned either on a concave buddle or sometimes on a knife buddle. The heads are next "tossed" in a kieve, and in this stage are known as "whits." The whits are next calcined, a rotating automatic calcining furnace being usually employed. Two types are in favour in Cornwall, namely Brunton's furnace, circular in form, revolving about a vertical axis, and Oxland and Hocking's, tubular in form, revolving about an axis slightly inclined to the horizontal. The object in either case is the decomposition of the arsenical and iron pyrites present, the sulphurous acid and arsenious acid passing off, the latter to be deposited and collected in special flues and chambers. Any copper present is converted into soluble sulphate which is leached out and precipitated by scrap iron. The iron present in the pyrites is converted into peroxide in a porous form in which it is easily washed away. The calcined whits are therefore again buddled and tossed, sometimes two or three times over, until the product now known as "black tin" is sufficiently clean for the smelter. If this black tin contains wolfram it can be cleaned by Oxland's process which consists in fusing it with black-ash (crude carbonate of soda) in a flat-bottomed reverberatory furnace. The oxide of tin is but little attacked, but the wolfram is decomposed, forming soluble sodic tungstate and spongy ferric oxide which is easily washed away. The final result is thus black tin containing not more than 5 per cent. of impurity. The slimes escaping at every stage of this process are run over tables or frames of various kinds, the self-acting Cornish frame being largely used. Any concentrates thus saved are cleaned by the same methods as are applied to the coarser stuff. A considerable proportion of fine slimed tin ore escapes even this further operation, and some of it is often caught on large canvas tables.

The old process was slow, cumbersome, expensive and wasteful, and has now been to a great extent abandoned. Works have been built identical in principle with those described above for treating copper ores. Thus a plant built by Messrs Fraser and Chalmers, Ltd. for treating a coarse-grained comparatively pure tin ore is shewn in Fig. 4[illegible]; it consisted of a Blake rock-breaker 15 inches by 9 inches breaking to 2½ inches. The broken ore goes to a trommel with 1¼ inch perforations, the oversize going to a 10 inch by 7 inch rock-breaker, and the undersize to a pair of 30 inch by 16 inch roughing rolls, which crush the ore to about ¾ inch. This crushed ore goes to a trommel with ⅝ inch holes, the oversize being further crushed in a pair of 26 inch by 15 inch rolls, whilst the undersize together with the re-crushed material goes to the sizing trommels, of which there are two sets of three with respectively ⅜ inch, ¼ inch and ⅛ inch perforations. The oversize from the first pair of trommels goes to a pair of 2-compartment jigs, from the second pair to a pair of 3-compartment and from the last pair of trommels to a pair of 4-compartment jigs, all the mill below the crushing rolls being built in two identical divisions. Only the tailings of the first pair of jigs are clean enough to be run to waste, all the others are re-crushed in a pair of Huntingdon mills, whilst the concentrates are clean enough for metallurgical treatment.

The undersize from the last trommel goes to a spitzkasten, the spigot delivering to a 4-compartment jig, and the overflow to other classifiers, which supply two more 4-compartment jigs, the tailings from the jigs going in every case to the Huntingdon mills. The re-crushed material goes to a large spitzkasten, the first spigot feeding two other 4-compartment jigs, and the remainder going to Frue vanners. Settling pits are provided for catching any heavy material that may escape with the tailings.

This mill was intended to treat 25 tons of ore per hour, with a power consumption of 127 I.H.P.

In some modern Cornish plants machinery of a more scientific type has also been introduced. Thus in one case[1] Californian stamps are used, running at 85 to 90 drops per minute, the screens being No. 37

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[1] *Trans. Inst. M. M.*, "On Crushing and Concentration at Dolcoath Mine, Cornwall," by R. A. Thomas, Vol. VII. 1899, p. 175.

hole punched copper sheets. The stamps crush at the rate of 1½ tons per 24 hours. The pulp goes to 27 Frue vanners, the concentrates from which go to the calciners and are then further washed. The tailings run to classifiers, the coarser portion being submitted to concentration and re-grinding, the finer being concentrated on revolving Harz tables.

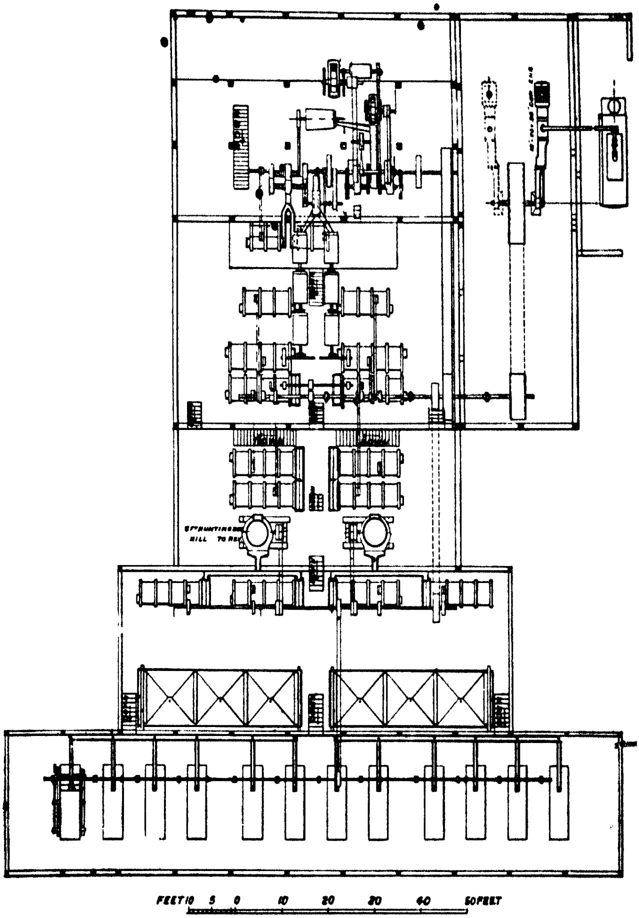

Fig. 411. Tin dressing plant.

In some cases the coarser portions of the crushed tin stuff is being treated on high speed jigs, and the finer portions on tables of the Wilfley type, vanners and revolving tables.

As an example of a modern plant, one recently erected in Cornwall[1] and shewn in Fig. 412 may be quoted. The bulk of the ore to be treated is tipped on to a pair of grizzlies with bars spaced two inches apart, the oversize going to a 15 inch by 10 inch Blake-Marsden rock-breaker, the broken stuff falling into the same hopper as the undersize from the grizzlies. The broken ore is fed by Challenge feeders into the four boxes of a battery of 20 Californian stamps, with 800 lbs. heads, making 95 drops per minute, and using No. 25 mesh gun-metal woven wire screens. The remainder of the ore is screened on a shaking screen, the oversize broken in a 12 inch by 8 inch rock-breaker, whence it passes to a pair of rolls 28 inches in diameter by 12 inches face, and thence to a No. 6 Krupp ball mill which crushes it to a 30 mesh screen. The pulp from the stamps and ball mill traverses three spitzlutten, two with three compartments and the third with two compartments. These eight spigots feed eight Buss tables (see p. 366) whilst the overflow goes to a large spitzkasten in 10 compartments, all of equal size, the area of the top being 50 feet by 6 feet, and the depth 5 feet 6 inches. The spigots of six of these compartments feed in pairs three distributing boxes, which supply three double Lührig vanners. The remaining four compartments feed a fourth pair of Lührig vanners. The overflow from this large spitzkasten and the tailings from the tables and vanners is collected in a similar spitzkasten but only in eight compartments, and thus 40 feet long, 6 feet wide and 5 feet 6 inches deep. The middlings from the machines are re-crushed in a wet ball mill and dressed on a pair of Lührig vanners. The concentrates from the Buss tables go direct to the calciner which appears to be of the old-fashioned Brunton type. The concentrates from the vanners are calcined separately, the light oxide of iron formed by the calcination is washed out on ordinary old-fashioned Cornish buddles, and the cleaned concentrates are dried. (Mr Dietsch points out that this buddling is not a desirable feature, but is adopted because the magnetic separator plant is not able to deal with all the concentrates produced.) The dry products go to a Wetherill magnetic separator (see p. 430) of the cross belt type having four pairs of poles. The first one takes out magnetic oxide of iron, the second other oxides with adherent oxide of copper, carrying at times up to 10 per cent. of copper, the

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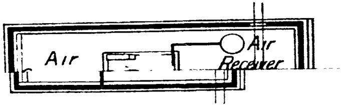
[1] *Inst. Min. Met.*, "The Treatment of Tin, Wolfram and Copper Ores at the Clitters United Mines," by F. Dietsch, Vol. xv. 1905, p. 2.



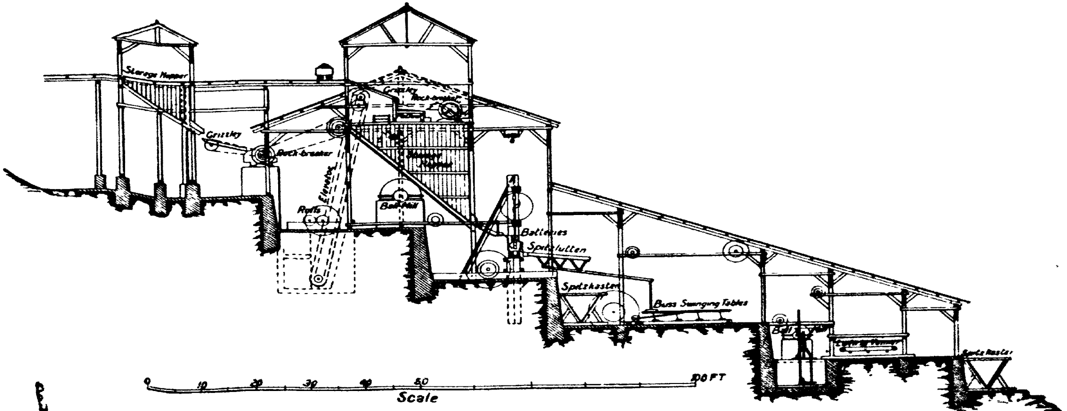


Fig. 412. Tin dressing plant.

third oxide of iron and wolfram, and the fourth tolerably clean wolfram. The last two products need only tossing in kieves to produce clean wolfram containing 12·5 per cent. of WO_3. The non-magnetic portion is also finished in tossing kieves and yields merchantable black tin containing 65 per cent. of SnO_2. The percentage of tungstic acid and stannic oxide together in the crude ore is given as 1·32 per cent. and the joint recovery as 1·17 per cent., equal to a recovery of 88·6 per cent. of these values. The capacity of the plant is about 100 tons of crude ore per 24 hours, and the working cost of treatment is given as 2*s.* 8·6*d.* per ton divided as follows:

Labour	15·90*d.* per ton.
Coal	12·03 ,,
Stores and repairs	4·67 ,,
Total	32·60*d.* per ton.

A modern plant for the treatment of tin ore in Devonshire is arranged as follows, the ore being quartzose and chloritic, containing 15 to 20 per cent. of arsenic as arsenical pyrites and 30 to 32 lbs. of black tin to the ton. The ore is broken in a rock-breaker and fed to two Husband's pneumatic stamps. These crush 20 to 22 tons per head per 24 hours; they are driven by electric motor, and consume about 32 H.P. when the shoes are new and 26 H.P. when worn, the average power consumption being 28 H.P. per head. They crush to 37 mesh Cornish, equal to about 28 holes to the linear inch. The pulp is sized in spitzkasten; the coarser spigots deliver onto 3 Buss tables, and the finer to 2 Lührig vanners. The overflow from this spitzkasten goes to a longer one that acts as a slime box, and delivers practically all the slimes to four concave revolving tables on the Harz principle (see p. 326). The middlings from the Buss tables are re-crushed and go to another Buss table. The whole of the dressing plant takes about 10 H.P. and is driven electrically. The concentrates consisting of arsenical pyrites, some iron pyrites and tinstone, are calcined on 2 fifteen foot Brunton calciners[1], the arsenious acid being collected in the usual system of flues. The burnt material is fed to a Wetherill magnetic separator of the cross-belt type, with two magnets (4 poles) in order to take out most of the oxide of iron, which carries with it very little tin (about 5 lbs. of black tin to the ton). The residue is then pulped with water and concentrated on a Buss table; the slimes are collected o[illegible] round Cornish buddle for further treatment; the middlings are ground and concentrated on round Cornish buddles. All the tinstone is finally cleaned by hand-tossing (see p. 253) and is then ready for the market. The total recovery of tin is said to be 90 per cent of that present in the crude ore. The whole plant is driven electrically, the current (100 ampères at 500 volts) being generated by a Campbell suction gas plant.

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[1] For a description of these see *The Handbook of Metallurgy*, Schnabel and Louis.

Silver ores. It may be said that silver ores proper (argentite, proustite, pyrargyrite, etc.) are never the subject of dressing operations. They may be submitted to grinding in the arrastra as a preliminary to amalgamation, or to dry-stamping in the Californian stamp mill as a preliminary to chloridising, etc., but these operations are better looked upon as a portion of metallurgical treatment, and as such will be found described in text books treating of this subject[1]; at the same time it may be pointed out that many of the true silver ores are so excessively brittle that they yield an excessive proportion of very fine slimes when pulverised, and hence such great losses would be incurred in any ordinary method of dressing as to render the operation quite unprofitable.

Argentiferous copper ores containing, e.g., grey copper ore, tetrahedrite, etc., may at times be concentrated, but these ores, too, are very brittle and excessive losses would have to be carefully guarded against. It is possible that the Elmore vacuum process or some analogous process may be employed with advantage in such cases. The general method would be like that described under the head of copper ores, but even greater care will have to be taken to re-treat middlings and tailings, the greater value of the material to be saved rendering this additional care economically admissible.

Argentiferous lead ores and zinc ores are at times concentrated by the methods described under lead and zinc ores; the clean separation of lead and zinc ores from each other is even more important when one or other or both are argentiferous than when they are poor in silver because any imperfection in the separation—particularly the presence of any notable proportion of zinc in the dressed lead ore—will entail heavy losses of silver in the subsequent metallurgical treatment. It is very often the case that ores of suitable composition are smelted direct without any previous concentration.

Gold ores. Here again the dressing operations are so closely interwoven with the metallurgical treatment that the subject is better dealt with by the metallurgist[2], amalgamation being, strictly speaking, a metal-

[1] See e.g. *Handbook of Metallurgy*, Schnabel and Louis, Vol. I.

[2] See e.g. *A Handbook of Gold Milling*, H. Louis.

lurgical operation. Alluvial deposits are treated by washing down in a current of water—which may be obtained in various ways, the coarse pulp thus produced passing over grizzlies to separate the boulders and larger pebbles, and then traversing a series of long strakes and labyrinths (known as "under-currents") in which the heavy particles of gold are allowed to deposit. They are practically always collected by means of mercury, which forms with them a heavy amalgam easily separable from the pulp. The finest particles of gold or amalgam are occasionally caught on canvas tables.

For gold ores, properly speaking, fine crushing is always required. A typical small mill with 20 heads of stamps is shewn in Fig. 413. The ore comes from the mine in cars, and is tipped over a grizzley *A* with bars $1\frac{1}{2}$ inches to 2 inches apart; the undersize drops into the main bin *B*, whilst the oversize goes to the rock-breaker *C*, set to break down to about $1\frac{1}{2}$ inch cubes. The broken ore drops into the same bin *B*, whence it runs through a shoot into Challenge feeders *D*, there being one to each battery of five stamps. The crushed ore passes out of the battery boxes *E* with water in the form of pulp, after, in many cases, the bulk of the free gold has been removed by amalgamation inside the battery box. The free gold left in the pulp is caught by streaming over plates *F* of amalgamated copper 6 to 12 feet long. Thence the pulp flows in launders down to the vanner room, where any pyrites or heavy minerals present are concentrated on Frue vanners *G*, *G'*. These concentrates are submitted to further treatment for the recovery of any gold they contain, this being a purely metallurgical operation. Very often the entire process of concentration is dispensed with, and the tailings (separated sometimes by spitzkasten into sands and slimes) are treated metallurgically, e.g. by the cyanide process.

Such a 20 stamp mill will treat from 50 to 100 tons of ore per 24 hours, with 3 or 4 men per shift. The horse-power required will be about 60 H.P. Such a stamp mill complete with buildings, engine and boilers would cost from £2500 to £3500 according to circumstances. It is generally preferable to work the rock-breaker by an independent engine, and in larger mills a separate engine is employed to drive the Frue vanner countershafts.

A 60 stamp mill for a gold mine in Tasmania laid out on somewhat similar lines, but rather more elaborate, is shewn in Fig. 414; the ore cars are tipped into bins, from the bottom of which an oscillating feeding tray discharges the ore into Gates rock-breakers, there being three of these in the mill; the broken stuff falls into three

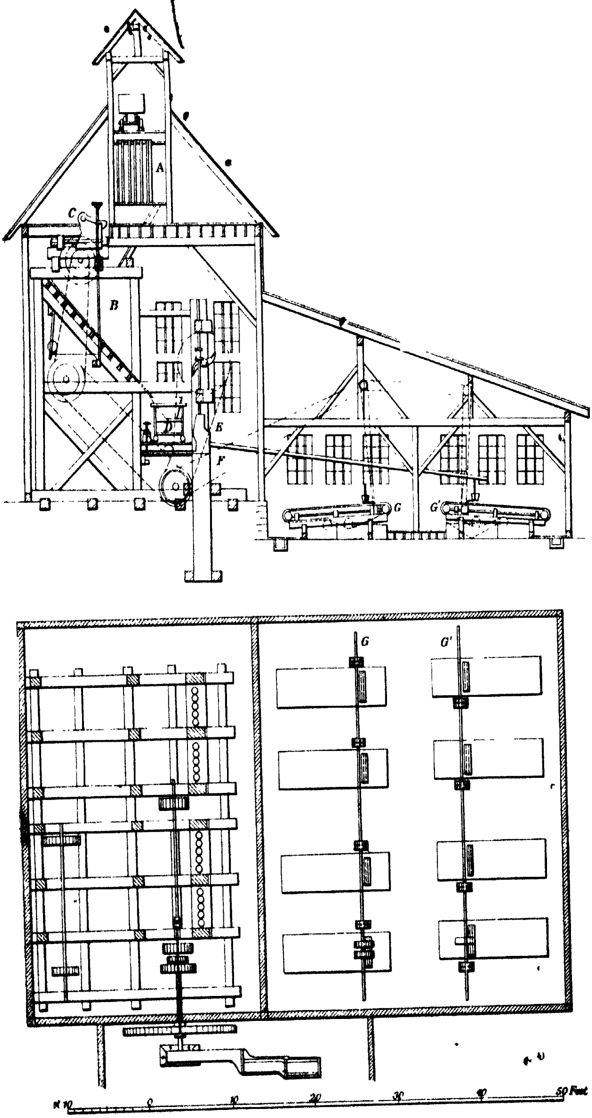

Fig. 413. Gold mill. Plan and sectional elevation.

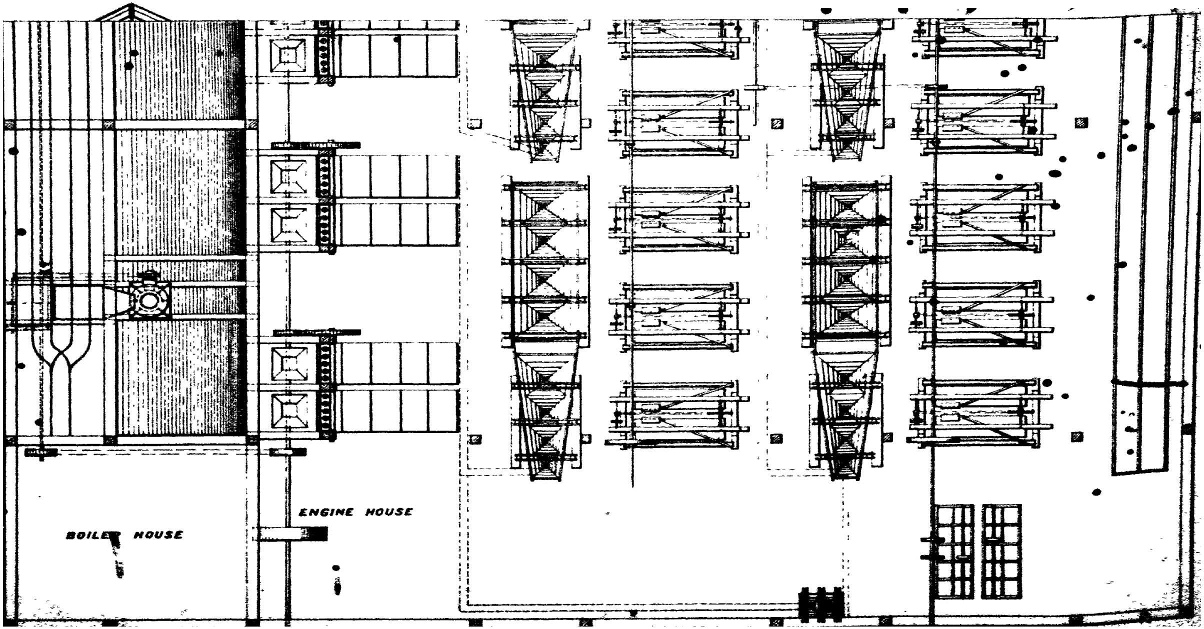

Fig. 414. GOLD MILL. Plan and Sectional Elevation.

storage bins, from which it is fed to the stamps by automatic feeders; each bin supplies four batteries or 20 stamps. After passing over the usual amalgamating tables, the pulp from each group of 20 stamps runs into spitzkasten; the spigots from the three spitzkasten supply pulp to 20 Lührig vanners arranged in pairs, back to back, and the overflow from the first three goes to a second group of three spitzkasten which supply other 20 Lührig vanners. It is obvious that this elaborate dressing plant could only be justified when the concentrates are rich in gold.

The general arrangement illustrated above in Fig. 413 is perhaps the best for small mills, and large mills are frequently built on exactly the same plan, consisting of a series of from 2 to 10 units such as the above. Very large mills of 80 stamps and more are often built in two lines, back to back, as shewn by the transverse section, Fig. 415[1], which shews the arrangement of a 120 head mill worked by steam power. This arrangement is a favourite one in the Witwatersrand district, where it is usual, however, to place the rock-breakers in a separate building, in which the ore is broken and sorted, this being the better arrangement for large mills. Very small mills used for prospecting or in the preliminary stages of opening up a mine may be equipped with a Huntingdon mill or Tremain steam stamps instead of the Californian stamps which are always employed for regular gold milling. When it is not intended to amalgamate the ore, but to treat it by chemico-metallurgical methods, comminution is sometimes performed by rolls or ball mills, followed in case of need by tube mills.

SALTS.

Very many salts, such as Chili nitre, common salt, some of the Stassfurth salts, etc., are purified by dissolving in water, and evaporating or crystallising the clarified solution. These are of the nature of chemical operations and will not be dealt with here. Other salts, e.g. gypsum and fluorspar are prepared for the market by simple hand-picking. Among the few salts to which true methods of dressing are applied may be mentioned barytes and apatite. In the trade the term barytes is applied, not only to the sulphate of baryta, but also to the carbonate, the mineralogical name for the latter, witherite, not being used. Both these minerals—which often occur together—are dressed by the methods used for lead ores, which the high specific gravity of

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[1] *Trans. Amer. Inst. Min. Eng.* Vol. XVII. 1888–9, p. 498.

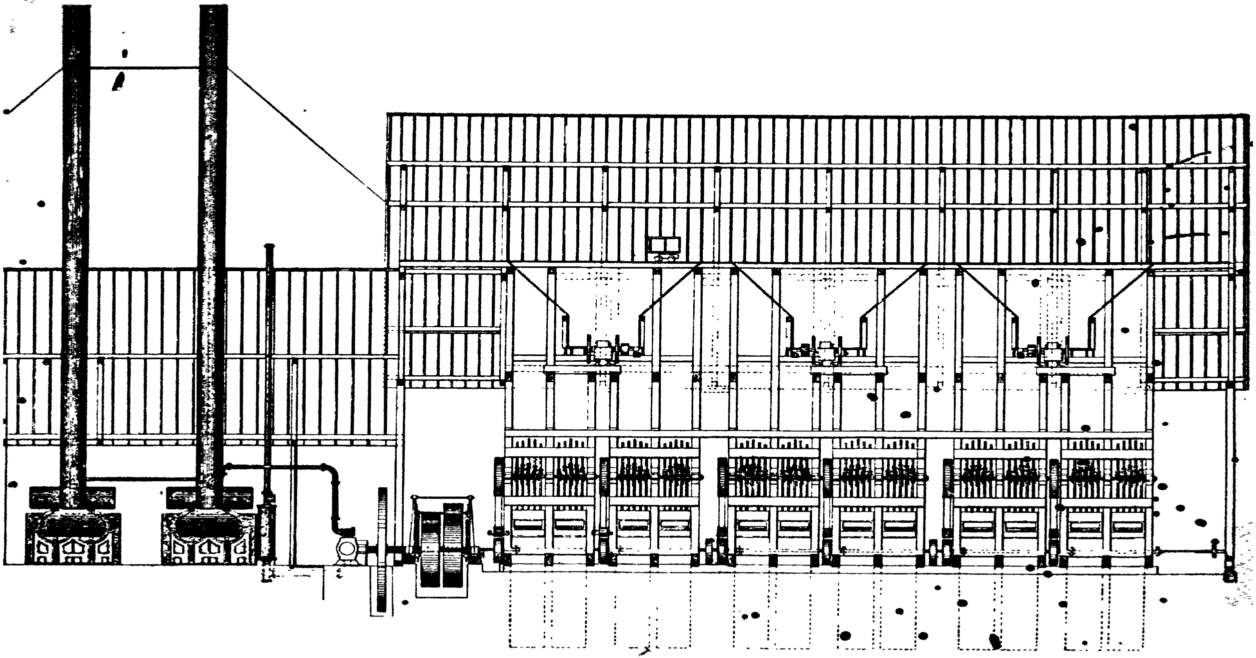


Fig. 415. 120 head gold mill. Longitudinal section.

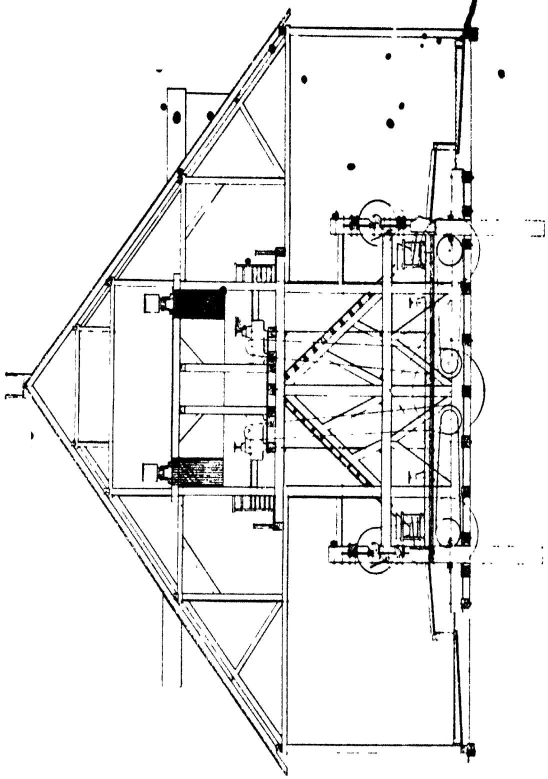


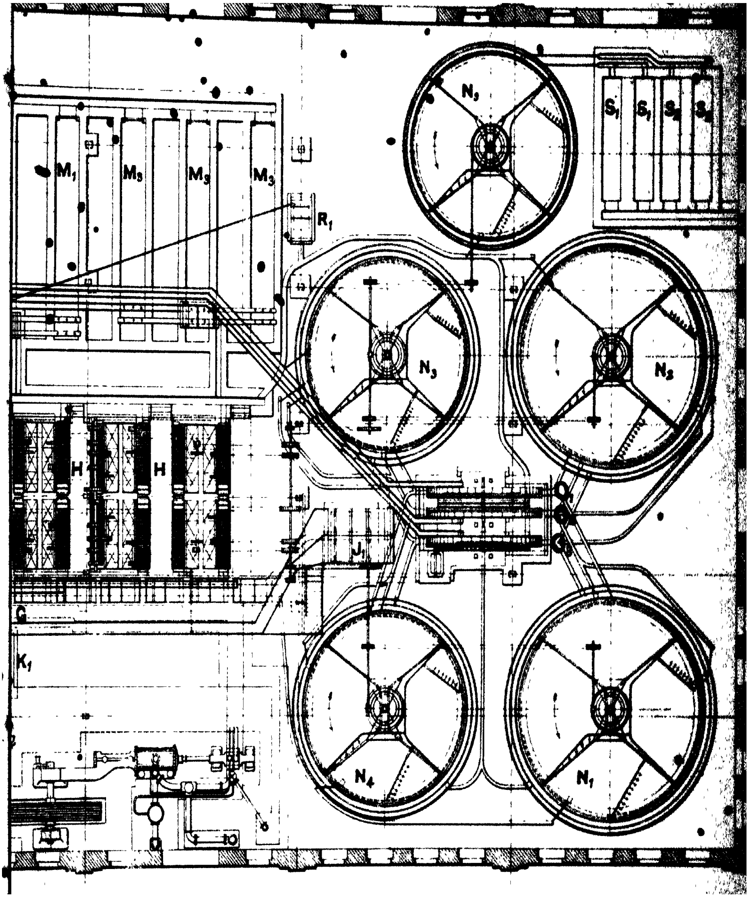
Fig. 415^a. 120 head gold mill. Transverse section.

the compounds of baryta renders entirely applicable. Usually the sulphate and carbonate of baryta are separated from each other as completely as possible by hand-picking, and then each mineral is dressed by itself. The crude mineral after breaking in a rock-breaker is hand-picked over a grizzley where lump mineral is picked out as completely as possible; the remainder goes to rough and fine rolls, is sized in trommels and each size jigged in a four-compartment Harz jig; any galena that may be present will be found in the first compartment, and this and the second compartment give clean barytes, the second and third compartments give material for re-crushing, and the tailings are usually not worth further treatment. The undersize from the trommels goes to a small classifier and thence usually to settling pits, the material from these being treated on buddles or similar appliances. The plants are for the most part small; the dressed mineral is often finely ground before it is sent to market. It is usually dressed to contain from 75 to 90 per cent. of the pure mineral, the selected lumps containing up to 98 per cent.

A good example of a plant for dressing phosphate of lime is a plant erected in Belgium by the Humboldt Engineering Co. and shewn in Fig. 416. The crude material is a soft earthy phosphate of lime, mixed with soft chalk and numerous flints. The crude mineral is tipped from a tippler **A** on to a screen **B** of the Briart type with bars $2\frac{3}{8}$ inches apart. The oversize falls down a shoot, where the larger flints are picked out, to a rock-breaker **C**. The undersize and the broken material pass to three disintegrators $\mathbf{D}_1$, $\mathbf{D}_2$, $\mathbf{D}_3$, which reduce the ore to $\frac{3}{8}$ inch, everything except the flints being ground to fine powder. The latter is mixed with water to a pulp, and lifted by an elevator wheel **E**; from this point the works are divided into two equal units, one on the right, the other on the left. The pulp goes to trommels **F** with $\frac{1}{16}$ inch mesh, the oversize from which is looked upon as waste and is removed. The undersize passes through the trough classifier **G**, the spigots of which supply 10 rows of three-compartment jigs, **H**, on either side; the tailings are elevated by centrifugal pumps, **K**, and flow to waste. The three compartments make each a separate grade of phosphate, each of which is elevated by one of the three wheels $\mathbf{L}_1$, $\mathbf{L}_2$, $\mathbf{L}_3$, which serve both sides of the mill, and flows through launders into twelve settling pits $\mathbf{M}_1$, $\mathbf{M}_2$, $\mathbf{M}_3$, there being six pits for the first grade, three for the second, and three for the third.

The overflow from the trough classifiers **G** flows to spitzkasten $\mathbf{J}_1$, $\mathbf{J}_2$, the spigots supplying Linkenbach tables $\mathbf{N}_1$—$\mathbf{N}_8$, the products from the first spigots going to tables 23 feet in diameter, and from the others to tables 26 feet in diameter. These tables make three classes of product, each of which is lifted by its own elevating wheel, $\mathbf{O}_1$—$\mathbf{O}_6$, into launders which deposit it in the settling pits $\mathbf{P}_1$, $\mathbf{P}_2$, $\mathbf{P}_3$. The tailings from these tables are also got rid of by the centrifugal pumps $\mathbf{K}_1$, $\mathbf{K}_2$.

The overflow from all the settling pits flows to a pump **Q** which sends it to two spitzkasten $\mathbf{R}_1$, $\mathbf{R}_2$, each with one spigot, supplying the two 23 feet Linkenbach tables $\mathbf{N}_9$, $\mathbf{N}_{10}$ making products which are settled in the pits $\mathbf{S}_1$, $\mathbf{S}_2$. The tailings from the Linkenbach tables flow to the waste pumps $\mathbf{K}_1$, $\mathbf{K}_2$, and the overflow from the last settling pits also runs to waste. The deposits in all the pits are allowed to settle firmly, are dug out and dried, and then form finished products, containing


N5
S1
S1
S2
S2
M1
M3
M3
M3
R1
N3
N2
H
H
J1
G
K1
N4
N1

40 to 50 per cent. of tribasic calcic phosphate, which is about double the percentage contained in the crude mineral. The plant can treat 250 tons of crude material in 10 hours and requires about 120 H.P. to work it.

Nodules of phosphate of lime often occur in tough clay and then require merely washing in a log-washer or some analogous appliance.

GEMS.

The great majority of gems occur in alluvial deposits from which they are recovered by screening and washing, usually by hand, followed by careful hand-picking. The washing is often performed on a crude hand-sieve.

As an example of the systematic dressing of gems, the treatment of diamondiferous "blue-ground" in South Africa may be quoted. The rock is exposed to the effects of air and moisture until it is weathered or disintegrated. This is then elevated and tipped into an iron shoot whence it is washed by jets of muddy water into trommels with about 1-inch mesh; the oversize, known as "lumps" or "cylinder-lumps," is returned to the depositing floors for further weathering. The pulp passes at the rate of 30 to 40 tons of dry material per hour, into washing pans (see p. 251), where the heavier portions are deposited, the tailings passing into another pan so as to catch any valuable material that may have escaped. The tailings are elevated on to heaps, so arranged that the greater part of the water in them drains back and is used over again for washing down the crude material, the separation being more effectual in muddy than in clean water. The heavy material deposited in the pans is drawn off at intervals into locked trucks, and is conveyed to the pulsator plant; it now amounts to less than 1 per cent. by volume of the original material. In the pulsator building it passes through a compound trommel with holes of ⅛ inch, 3/16 inch, ¼ inch and ⅜ inch in diameter. The oversize from the trommels drops on to a sorting table where any diamonds present are picked out. Each size is jigged separately in a single-compartment Harz jig, the bed being formed of leaden bullets. The tailings run to waste, and the contents of the hutches are sorted by hand. For the finer sizes a shaking table covered with a thick grease, to which diamonds adhere, is often employed. Of the material sent to the pulsator, 33 per cent. is larger than ⅜ inch and forms the oversize of the trommels, 8 per cent. forms hutchwork in the jigs, and 59 per cent. forms jig tailings. The concentrated material has to be repeatedly sorted, so as to prevent loss of diamonds.

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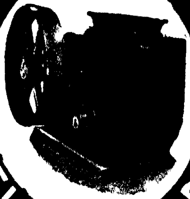

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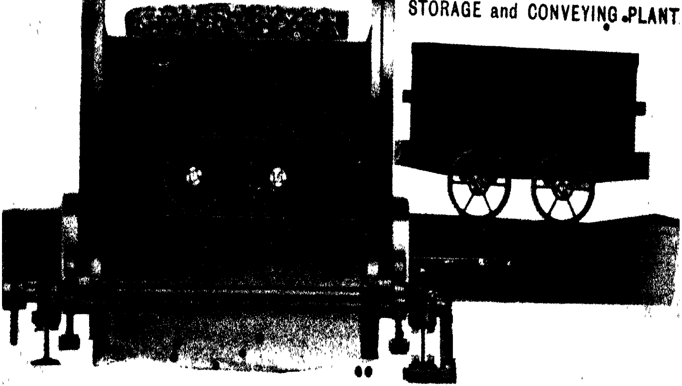

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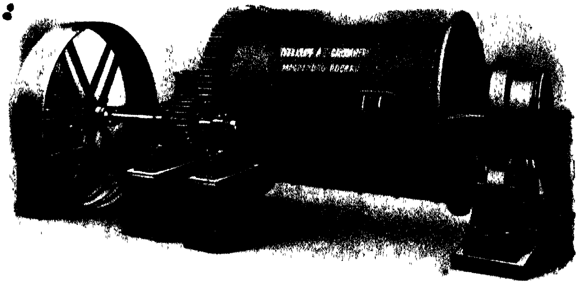

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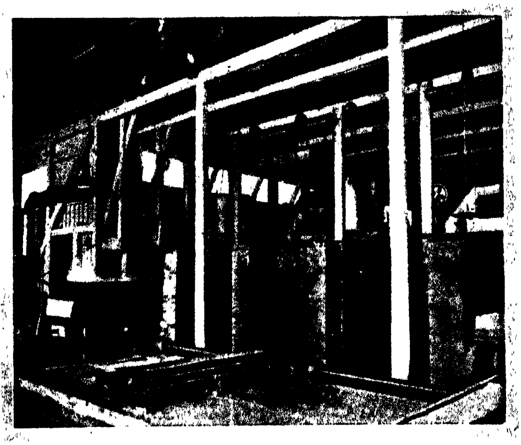

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